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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XVII.

MAY, 1888, TO FEBRUARY, 1889,

INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY
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SHERMAN & CO., PRINTERS.

PREFACE.

In consequence of the large space required in this volume for Proceedings and Papers, the list of members has been omitted. Such lists are published frequently in pamphlet form and distributed to members, and extra copies may be had at any time from the Secretary, on the terms stated in this volume under the head of Publications (page xii).

CONTENTS.

PAGE

OFFICERS, ... ix
LIST OF MEETINGS, ... x
PUBLICATIONS, ... xi
RULES, ... xiii

PROCEEDINGS.

Birmingham Meeting, May, 1888, ... xix
Buffalo Meeting, October, 1888, ... xxiv
New York (Annual) Meeting, February, 1889, ... xxxi

PAPERS.

The Losses in Roasting Gold-Ores and the Volatility of Gold. By SAMUEL B. CHRISTY, ... 3
The Cost of a Ton of Pig-Iron in the Sequachee Valley. By WILLIAM M. BOWRON, ... 45
The Efficiency of a Steam-Boiler using the Waste Gas of a Blast-Furnace as Fuel. By D. S. JACOBUS, ... 50
Henderson Steel. By ALFRED F. BRAINERD, ... 60
A Water-Manometer and Anemometer. By J. M. SILLIMAN, ... 66
Note on Arsenic Determination. By R. C. CANBY, ... 77
Calculations of the Available Heat and the Required Dimensions of Chimneys, Combustion-Chambers, and Gas-Burners in the Use of Blast-Furnace Gases for Firing Boilers. By FRANK C. ROBERTS, ... 78
Phosphate-Slag. By WILLIAM B. PHILLIPS, ... 84
The Grading of Birmingham Pig-Iron. By KENNETH ROBERTSON, . . . 94
The Feasibility of using Cheaper Fuels in the Blast-Furnace. By JACOB T. WAINWRIGHT, ... 96
Determination of Phosphorus in Iron and Steel. By PORTER W. SHIMER, . 100
Mining in Soft Ore-Bodies at Low Moor. By W. S. HUNGERFORD, . . 103
The Petite Anse Salt-Mine. By RICHARD A. POMEROY, ... 107
Note on a Specimen of Gilsonite from Uintah County, Utah. By R. W. RAYMOND, ... 113
The Glenmore Iron Estate, Greenbrier County, West Virginia. By WILLIAM N. PAGE, ... 115
Anthracite and Coke, Separate and Mixed, in the Warwick Blast-Furnace. By EDGAR S. COOK, ... 124
The Flue-Dust of the Furnaces at Low Moor, Virginia. By ELLISON C. MEANS, ... 129
Note on Cast-Steel Water-Jackets. By RICHARD H. TERHUNE, ... 131
A Differential Regenerative Hot-Blast Stove and its Application to an Open-Hearth Blast-Furnace. By JACOB T. WAINWRIGHT, ... 132
Large Furnaces on Alabama Material. By FRED. W. GORDON, ... 135
Note on the Iron-Ores, Fuels and Improved Blast-Furnace Practice of the Birmingham District. By ALFRED F. BRAINERD, ... 151
Notes on the Geology and on some of the Mines of Aspen Mountain, Pitkin County, Colorado. By CARL HENRICH, ... 156
The Development and Statistics of the Alabama Coal-Fields for 1887. By CHARLES A. ASHBURNER, ... 206
Steel Rails and Specifications for their Manufacture. By ROBERT W. HUNT, 226
Note on the Cultivation of Mushrooms in Abandoned Mines at Akron, New York. By WILLIAM Y. WARREN, ... 248
Cement-Rock and Gypsum Deposits in Buffalo. By JULIUS POHLMAN, . 250

PAGE

Ferro-Silicon and the Economy of its Use. By W. J. KEEP and EDWARD ORTON, JR., ... 253
Hot-Spring Formations in Red Mountain District, Colorado. A Reply to the Criticisms of Mr. Emmons. By THEODORE B. COMSTOCK. 261
The Mining Industry in its Relation to Forestry. By B. E. FERNOW, . . 264
Notes on the Roasting of the Hudson River Carbonates. By INGERSOLL OLMSTED, ... 275
The Effect of Velocity and Tension of Gases on the Reduction of Ores in the Blast-Furnace. By THEO. W. ROBINSON, ... 282
The Minerals of Ontario and their Development. By WILLIAM HAMILTON MERRITT, ... 293
Water-Gas as a Steam-Boiler Fuel. By D. S. JACOBUS, ... 300
The Equalization of Load on Winding-Engines by the Employment of Spiral Drums. By E. M. ROGERS, ... 305
The Chlorination of Low-Grade Auriferous Sulphides. By WILLIAM B. PHILLIPS, ... 313
The Life-History of Niagara. By JULIUS POHLMAN, ... 322
The Impurities of Water. By A. E. HUNT and G. H. CLAPP, ... 338
Asphalt and its Uses. By F. V. GREENE, ... 355
The Northwestern Colorado Coal-Region. By G. C. HEWETT, ... 375
A Present Need in the Engineering Profession. By WILLIAM B. POTTER, . 380
Tuyere Slagging-Valve. By EDGAR S. COOK, ... 389
The Geology of Buffalo as Related to Natural-Gas Explorations along the Niagara River. By CHARLES ALBERT ASHBURNER, ... 398
Notes on the Electrolytic Assay of Copper. By WILLIAM GLENN, . . 406
A Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis. (See Discussion, p. 757.) By CLEMENS JONES. 411
Biographical Notice of Erich C. Schaufuss. By J. H. BOWDEN, . . . 419
Rail-Sections. By FREDERIC A. DELANO, ... 421
Hollow Iron Pig-Patterns. By B. F. FACKENTHAL, JR, ... 427
Note on the Koepe System of Winding from Shafts. By JOHN H. HARDEN, 429
Notes on the Rosario Mine at San Juancito, Honduras, C. A. By THOMAS H. LEGGETT, ... 432
Soaping Geysers. By R. W. RAYMOND, ... 449
Coal-Transfer of the Mt. Carbon Company, Limited. By W. N. PAGE, . 454
Pig-Iron of Unusual Strength. By FRED. P. DEWEY, ... 460
Note upon some Results of the Storage of Water in Arizona. By WILLIAM P. BLAKE, ... 476
The Copper-Deposits of Copper Basin, Arizona, and their Origin. By WILLIAM P. BLAKE, ... 479
Some Thoughts Relating to the American Institute of Mining Engineers and Its Mission. By WILLIAM B. POTTER, ... 485
Nails from Tin-Scrap. By OBERLIN SMITH, ... 495
Gold-Milling in the Black Hills. By H. O. HOFMAN, ... 498
The Determination of Silicon in Ferro-Silicons; Its Occurrence in Aluminum as Graphitoidal Silicon; and a study of Its Reactions with Alkaline Carbonates. By HENRY J. WILLIAMS, ... 542
Soaping Geysers. By ARNOLD HAGUE, ... 546
The Present Status of Electric Transmission of Power. By RICHARD P. ROTHWELL, ... 555
Automatic Dumping-Cradles for Mine Cars. By H. S. MUNROE, . . . 564
Ore-Deposits of the Black Hills of Dakota. By FRANKLIN R. CARPENTER, . 570
The Wenström Magnetic Separator. By ROBERT ANDERSON COOK, . . 599
Notes on the Bernice Anthracite Coal-Basin, Sullivan County, Pa. By CLARENCE R. CLAGHORN, ... 606
The Distribution of Phosphorus in the Ludington Mine, Iron Mountain, Michigan; A Study in Isochemic Lines. By DAVID H. BROWNE, . 616
Note on the Influence of Columbite on the Tin-Assay. (See Discussion, p. 785). By FRANKLIN R. CARPENTER and W. P. HEADDEN, . . . 633
Note on the Nickel-Ore of Russell Springs, Logan County, Kansas. By FRED. P. DEWEY, ... 636
The English *versus* the Continental System of Jigging—Is Close Sizing Advantageous? By H. S. MUNROE, ... 637
The New Dressing-Works of the St. Joseph Lead Company at Bonne Terre, Missouri. By H. S. MUNROE, ... 659

PAGE

A Note upon a Modification of the Reducing Process Used by the Carbon Iron Company. By ALFRED E. HUNT, ... 678
A New System for Operating Regenerative Hot-Blast Stoves. By JACOB T. WAINWRIGHT, ... 680
Silicon in Cast-Iron. By W. J. KEEP (Analytical Determinations by H. S. FLEMING and EDWARD ORTON, JR.), ... 683
Prominent Sources of Iron-Ore Supply. By JOHN BIRKINBINE, ... 715
The Concentration of Iron-Ore. By JOHN BIRKINBINE and THOMAS A. EDISON, ... 728
A Review of the Iron-Mining Industry of New York for the past Decade. By JOHN C. SMOCK, ... 745
A Rapid Method for the Determination of Phosphorus in Certain Ores. By T. REED WOODBRIDGE, ... 750
Supplementary Note on Blast-Furnace Lines. By EDWARD WALSH, Jr., . 754
A Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis. (Discussion of Paper on p. 411), ... 757
The Reopening of the Tilly Foster Iron-Mine. By F. H. MCDOWELL, . 758
The Occurrence and Treatment of the Argentiferous Manganese Ores of Tombstone District, Arizona. By CHARLES W. GOODALE, ... 767
Proposed Rail-Sections. By ROBERT W. HUNT, ... 778
Note on the Influence of Columbite on the Tin-Assay. (Discussion of Paper on p. 633). By W. P. BLAKE, ... 785
End-Lines and Side-Lines in the U. S. Mining Law. By R. W. RAYMOND, 787
Note on the Use of Crude Petroleum as Fuel for Raising Steam at the South Chicago Works. By E. C. POTTER, ... 807
Coal *versus* Oil in the Puddling-Furnace, and in Raising Steam. By G. H. BILLINGS, ... 808
Oil as a Metallurgical Fuel. By E. C. FELTON, ... 809

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* Richard Pearce, Argo, Colo., was elected President in February, 1889.

† Messrs. Eckley B. Coxe, Drifton, Pa.; Charles MacDonald, New York, N. Y., and Percival Roberts, Jr., Philadelphia, Pa., were elected Vice-Presidents in February, 1889.

‡ Messrs. J. H. Bramwell, Bramwell, W. Va.; Frank Firmstone, Easton, Pa., and William H. Pettee, Ann Arbor, Mich., were elected Managers in February 1889.

⁂ Dr. Percy died while the present volume was in press.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO JUNE, 1889.

Number.	Place.	Date.	Transactions.
I.	Wilkes-Barre, Pa.,*	May, 1871,	i. 3
II.	Bethlehem, Pa.,	August, 1871,	i. 10
III.	Troy, N. Y.,	November, 1871,	i. 13
IV.	Philadelphia, Pa.,	February, 1872,	i. 17
V.	New York, N. Y.,*	May, 1872,	i. 20
VI.	Pittsburgh, Pa.,	October, 1872,	i. 25
VII.	Boston, Mass.,	February, 1873,	i. 28
VIII.	Philadelphia, Pa.,*	May, 1873,	ii. 3
IX.	Easton, Pa.,	October, 1873,	ii. 7
X.	New York, N. Y.,	February, 1874,	ii. 11
XI.	St. Louis, Mo.,*	May, 1874,	iii. 3
XII.	Hazleton, Pa.,	October, 1874,	iii. 8
XIII.	New Haven, Conn.,	February, 1875,	iii. 15
XIV.	Dover, N. J.,*	May, 1875,	iv. 3
XV.	Cleveland, O.,	October, 1875,	iv. 9
XVI.	Washington, D. C.,	February, 1876,	iv. 18
XVII.	Philadelphia, Pa.,†	June, 1876,	v. 3
XVIII.	Philadelphia, Pa.,	October, 1876,	v. 19
XIX.	New York, N. Y.,	February, 1877,	v. 27
XX.	Wilkes-Barre, Pa.,*	May, 1877,	vi. 3
XXI.	Amenia, N. Y.,	October, 1877,	vi. 10
XXII.	Philadelphia, Pa.,	February, 1878,	vi. 18
XXIII.	Chattanooga, Tenn.,*	May, 1878,	vii. 3
XXIV.	Lake George, N. Y.,	October, 1878,	vii. 103
XXV.	Baltimore, Md.,*	February, 1879,	vii. 217
XXVI.	Pittsburgh, Pa.,	May, 1879,	viii. 3
XXVII.	Montreal, Canada,	September, 1879,	viii. 121
XXVIII.	New York, N. Y.,*	February, 1880,	viii. 275
XXIX.	Lake Superior, Mich.,	August, 1880,	ix. 1
XXX.	Philadelphia, Pa.,*	February, 1881,	ix. 275
XXXI.	Staunton, Va.,	May, 1881,	x. 1
XXXII.	Harrisburg, Pa.,	October, 1881,	x. 119
XXXIII.	Washington, D. C.,*	February, 1882,	x. 225
XXXIV.	Denver, Col.,	August, 1882,	xi. 1
XXXV.	Boston, Mass.,*	February, 1883,	xi. 217
XXXVI.	Roanoke, Va.,	June, 1883,	xii. 3
XXXVII.	Troy, N. Y.,	October, 1883,	xii. 175
XXXVIII.	Cincinnati, O.,*	February, 1884,	xii. 447
XXXIX.	Chicago, Ill.,	May, 1884,	xiii. 1
XL.	Philadelphia, Pa.,	September, 1884,	xiii. 285
XLI.	New York, N. Y.,*	February, 1885,	xiii. 585
XLII.	Chattanooga, Tenn.,	May, 1885,	xiv. 1
XLIII.	Halifax, N. S.,	September, 1885,	xiv. 307
XLIV.	Pittsburgh, Pa.,*	February, 1886,	xiv. 587
XLV.	Bethlehem, Pa.,	May, 1886,	xv. lxiii.
XLVI.	St. Louis, Mo.,	October, 1886,	xv. lxx.
XLVII.	Scranton, Pa.*	February, 1887,	xv. lxxvii.
XLVIII.	Utah and Montana,	July, 1887,	xvi. xvii.
XLIX.	Duluth, Minn.,	July, 1887,	xvi. xxiv.
L.	Boston, Mass.,*	February, 1888,	xvi. xxviii.
LI.	Birmingham, Ala.,	May, 1888,	xvii. xix.
LII.	Buffalo, N. Y.,	October, 1888,	xvii. xxiv.
LIII.	New York, N. Y.,*	February, 1889,	xvii. xxxi.
LIV.	Denver and Aspen, Col.,	June, 1889,	xviii.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

1. The minutes of the Proceedings of each Meeting, published in pamphlet form.

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RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, MAY, 1877, MAY, 1878, FEBRUARY, 1880, FEBRUARY, 1881, FEBRUARY, 1887, AND FEBRUARY, 1889.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance at the annual meeting. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

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V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Two other regular meetings of the Institute shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these rules from and after the publication of said announcement by the Secretary.

PROCEEDINGS.

PROCEEDINGS OF THE FIFTY-FIRST MEETING, BIRMINGHAM, ALABAMA, MAY, 1888.

LOCAL COMMITTEE.

Kenneth Robertson, *Chairman;* S. P. Carter, Jr., *Treasurer;* W. H. Woodward, E. M. Tutwiler, W. H. Hassinger, Thomas Seddon, T. H. Aldrich, H. W. Hargraves, E. Thomas, W. B. Foote, George L. Morris, J. M. Searle, Thomas Ward, L. R. Brooks, T. A. Mack, T. T. Hillman, C. Cadle, Jr., F. Sloss, G. W. Ware, J. H. McCune, Giles Edwards, J. W. Bush, A. F. Brainerd, E. W. Rucker, A. B. Johnston, C. P. Williamson, J. B. Boddie, E. Eastman, W. T. Underwood, and A. G. Lane.

THE sessions of the meeting were held at the Grand Opera House.

The first session was held on Tuesday morning, May 15th. Mr. Kenneth Robertson, Chairman of the Local Committee, called the meeting to order, and introduced Mr. E. T. Taliaferro, who welcomed the Institute to Birmingham, in the name of the citizens generally, and was followed by Mr. James F. Webb, who extended a similar cordial greeting on behalf of the Chamber of Commerce.

After responses by President W. B. Potter and the Secretary, the following papers were read:

Biographical Notice of Byron W. Cheever, by W. H. Pettee, Ann Arbor, Mich. (Read by the Secretary, in the absence of the author. This notice was published in Vol. XVI., p. 888.)

Henderson Steel, by A. F. Brainerd, Birmingham, Ala.

The Cost of a Ton of Pig-Iron in the Sequachee Valley. By Wm. M. Bowron, South Pittsburgh, Tenn.

The second session, held on Tuesday evening, was devoted to the reading and discussion of a paper on Phosphate Slag, by Professor William B. Phillips, Chapel Hill, N. C.

The third session was held on Thursday morning, May 17th, when the following paper was read and discussed:

Large Furnaces on Alabama Material, by Fred. W. Gordon, Philadelphia, Pa. (Read by the Secretary, in the absence of the author.)

The following persons having been duly recommended by the Council, were unanimously elected:*

MEMBERS.

Walter Atherton,	Boston, Mass.
Rudolph A. Bergier,	Butte City, Montana.
Jesse C. Boyd,	Cornwall, Pa.
Edward E. Brewster,	Iron Mountain, Mich.
L. R. Brooks,	Birmingham, Ala.
Charles H. Cady,	Iron Mountain, Mich.
Lafayette D. Carroll,	Birmingham, Ala.
Elihu D. Church, Jr.,	Brooklyn, N. Y.
John Y. Colby, Jr.,	New York City.
Henry D. Conant,	Opechee, Mich.
S. B. Connor,	San Francisco, Cal.
William C. Crozer,	Knoxville, Tenn.
Paul Daleas,	Paris, France.
Charles J. Develin,	Spring Valley, Ill.
Enoch Ensley,	Ensley City, Ala.
John B. Farish,	Denver, Col.
George D. Fitzhugh,	Birmingham, Ala.
D. M. Forker,	Birmingham, Ala.
A. G. Francis,	Corona, Ala.
John H. Fricke,	Kingston, New Mexico.
Cyrus O. Godfrey,	Fort Payne, Ala.
Marcus Grant,	Chattanooga, Tenn.
William Hainsworth,	Pittsburgh, Pa.
Allen W. Haskell,	Blocton, Ala.
Nathaniel Hathaway,	New Bedford, Mass.
Eugene Hoefer,	Pinal, Arizona.
John C. Jackson,	Chicago, Ill.
David S. Jacobus,	Hoboken, N. J.
J. M. Jaycox,	Steelton, Pa.
Leon Jousselin,	Medellin, Columbia.
Fred. C. Keighley,	Uniontown, Pa.
Masa Kuwabara,	Osaka, Japan.
Olin H. Landreth,	Nashville, Tenn.
W. G. La Rue,	Bessemer, Mich.
John Lodge,	South Pittsburgh, Tenn.
William T. Magruder,	Nashville, Tenn.
Henry L. Manley,	Shamokin, Pa.
Henry McCalley,	University, Alabama.
John J. Mickley,	Hokendauqua, Pa.
A. D. Moffat,	Park City, Utah.
Robert Morrison,	Chattanooga, Tenn.
Walter Moore,	Birmingham, Ala.
A. Moulle,	Paris, France.

* This list includes the names of a few persons elected at the supplementary session at Anniston, Ala., on Saturday evening.

William Oliver,	Iron Mountain, Mich.
G. S. Patterson,	Anniston, Ala.
Erskine Ramsay,	Pratt Mines, Ala.
T. A. Rickard,	San Andreas, Cal.
John E. Rothwell,	Glenwood, Mass.
J. R. Ryan,	Birmingham, Ala.
Thomas I. Scott,	St. Louis, Mo.
Earle Sloan,	Birmingham, Ala.
Montagu Soilleux,	London, England.
Walter M. Stein,	Philadelphia, Pa.
George D. Stonestreet,	Birmingham, Ala.
Joseph Struthers, Jr.,	New York City.
H. B. Sturtevant,	Ishpeming, Mich.
Peter B. Thomas,	Blocton, Ala.
James R. Thompson,	Negaunee, Mich.
Henry B. Tompkins,	Atlanta, Ga.
Priestley Toulmin,	Coalburg, Ala.
Jacob T. Wainwright,	Pittsburgh, Pa.
C. P. Williamson,	Birmingham, Ala.

ASSOCIATES.

Nelson S. Bartlett,	Boston, Mass.
John W. Bush,	Birmingham, Ala.
Charles P. Coleman,	South Bethlehem, Pa.
Karl Eilers,	Brooklyn, N. Y.
Harry W. English,	Birmingham, Ala.
George Warren Hammond,	Yarmouthville, Me.
Gilmer Meriwether,	Birmingham, Ala.
William R. Miller,	Baltimore, Md.
Henry B. C. Nitze,	South Bethlehem, Pa.
Clement Stewart,	Easton, Pa.
Harry H. Stoek,	Knowles, Md.
D. P. West,	Birmingham, Ala.
A. P. Wilcox,	St. Louis, Mo.

Mr. R. M. Edwards, associate, was, on recommendation of the Council, made a member.

The following papers were read by title:

Calculations of the Available Heat and the Required Dimensions of Chimneys, Combustion-Chambers, and Gas-Burners in the Use of Blast-Furnace Gases for Heating Boilers, by F. C. Roberts, Philadelphia, Pa.

Note on Arsenic Determinations, by R. C. Canby, Leadville, Col.

Determination of Phosphorus in Iron and Steel, by Porter W. Shimer, Easton, Pa.

The Development and Statistics of the Alabama Coal-Fields for 1887, by Charles A. Ashburner, Pittsburgh, Pa.

The Efficiency of a Steam-Boiler Using the Waste Gas of a Blast-Furnace as Fuel, by D. S. Jacobus, Hoboken, N. J.

The Petite Anse Salt-Mine, by Richard A. Pomeroy, New Iberia, La.

The Feasibility of Using Cheaper Fuels in the Blast-Furnace, by Jacob T. Wainwright, Pittsburgh, Pa.

Notes on the Iron-Ores, Fuels and Improved Blast-Furnace Practice of the Birmingham District, by Alfred F. Brainerd, Birmingham, Ala.

The Grading of Birmingham Pig-Iron, by Kenneth Robertson, Birmingham, Ala.

Mining in Soft Ore-Bodies at Low Moor, by W. S. Hungerford, Low Moor, Va.

The Losses in Roasting Gold-Ores, and the Volatility of Gold, by S. B. Christy, Berkeley, Cal.

Notes on the Geology and on Some of the Mines of Aspen Mountain, Pitkin County, Colo., by Carl Henrich, Noble, Ill.

A Water-Manometer and Anemometer, by Prof. J. M. Silliman, Easton, Pa.

After the passage of a resolution, instructing the Secretary to express the thanks of the Institute to the local committee and the various corporations and citizens, for their hospitable reception of visiting members and guests, the meeting was adjourned.

Excursions and Entertainments.

Tuesday afternoon, May 15th, an excursion train on the Birmingham Mineral Railroad conveyed a large party to the Morris and other red fossil-ore mines on the south side of Red Mountain.

Wednesday was spent in an excursion over the Birmingham Mineral Railroad to the North Birmingham furnaces of the Sloss Company, the furnace of the Thomas Iron Company, the furnaces, coal-mines and coke-ovens of the Tenn., C. I. & R.R. Co., the furnaces of the De Bardeleben C. & I. Co., the Bessemer rolling-mills, etc., including a lunch at Ensley by invitation of the Ensley Land Company.

Thursday morning, after the session, a visit was made to the Henderson Steel Works.

Thursday afternoon, an excursion by the Alabama Great Southern Railroad visited the Gate City rolling-mill, ore-bank and limestone-quarry, and the Trussville furnace.

On Thursday evening, a subscription-dinner took place at the restaurant of the Alabama Club.

Friday was devoted to an excursion, leaving Birmingham by the Mineral Railroad, and visiting the Blue Creek coal-basin, the limonite deposit at Woodstock, and the Lower Cahaba coal-field at Blocton. The party was entertained at Blocton by the Cahaba Coal Mining Company.

Saturday morning, a special train on the Georgia Pacific Railroad conveyed the members and guests to Anniston, Ala., where, after lunch at the Anniston Inn, the works of the Woodstock Iron Company, and other establishments, including large pipe-works in progress, were visited. An informal session and social reception took place in the evening at the Anniston Inn.

On Monday morning, May 21st, an excursion was made by special train on the Anniston and Atlantic (narrow gauge) Railroad to the ore-banks at and beyond the Clifton furnace. Returning from this excursion about noon, most of the members and guests departed in various trains, North and East. Those who remained were entertained in the evening at a ball given in their honor at the Anniston Inn.

The excursions around Anniston were managed by a local committee, of which Mr. O. H. Parker was chairman.

Members and Associates Present.

The following members and associates were registered at the headquarters of the Local Committee, in Birmingham. No registry was kept at Anniston, where a few others joined the party.

T. H. Aldrich.
N. S. Bartlett.
William M. Bowron.
Alfred F. Brainerd.
T. B. Brooks.
C. Cadle, Jr.
John B Carrington.
L. D. Carroll.
S. P. Carter, Jr.
William H. Case.
H. S. Chamberlain.
C. R. Clagborn.
George W. Cope.
W. L. Dudley.
M. M. Duncan.
Hunter Eckert.
Giles Edwards.
R. K. Edwards.
Thomas J. Edwards.
Isaac Fegely.
C. B. Finley.
F. Firmstone.
W. B. Foote.
D. M. Forker.

R. C. Fulton.
W. H. Hassinger.
G. Jamme.
Joseph C. Kent.
J. S. Lane.
P. D. Langdon.
J. F. Lewis.
John Lodge.
Henry McCalley.
J. H. McCune.
J. King McLanahan.
Thomas A. Mack.
W. F. Mattes.
J. R. Maxwell.
John J. Mickley.
H. Key Milner.
George L. Morris.
William G. Neilson.
Edward Nichols,
Charles M. Noble.
S. E. Noble.
James J. Ormsbee.
I. P. Pardee.
C. O. Parsons.
G. S. Patterson.
E. C. Pechin.
W. B. Phillips.
S. H. Pitkin.
William B. Potter.
R. W. Raymond.
Jacob Reese.
G. Collier Robbins.
Kenneth Robertson.
C. L. Rogers.
John R. Ryan.
Thomas J. Scott.
Minor Scovel.
Thomas Seddon.
A. M. Shook.
Fred. Sloss.
Joseph Squire.
W. M. Stein.
George D. Stonestreet.
V. O. Strobel.
J. K. Surls.
William Thaw, Jr.
A. Thies.
D. H. Thomas.
E. Thomas.
Priestley Toulmin.
E. M. Tutwiler.
George W. Ware.
Francis E. Weston.
John F. Wilcox.
W. H. Wiley.
J. Frank Wilkes.
C. P. Williamson.
J. P. Witherow.
W. H. Woodward.

PROCEEDINGS OF THE FIFTY-SECOND MEETING, BUFFALO, N. Y., OCTOBER, 1888.

LOCAL COMMITTEE.

Edward Nichols, Dunkirk, N. Y., *Chairman;* T. Guilford Smith, Buffalo, N. Y. H. F. Gaskill, Lockport, N. Y., William Foster, Piffard, N. Y. and M. L. Hinman, Dunkirk, N. Y., *Vice-Chairmen;* C. D. R. Stowitz, Buffalo, N. Y., *Treasurer;* Dr. Julius Pohlman, Buffalo, N. Y., *Secretary;* R. B. Adam, James Archibald, John J. Albright, W. I. Babcock, Charles F. Bingham, T. A. Bissell, Henry W. Box, William Bookstaver, Dunkirk; Andrew Brown, W. C. Bryant, P. P. Burtis, John W. Cloud, W. C. Cowles, Lockport; J. H. Dawes, David F. Day, J. P. Dudley, E. T. Evans, C. M. Farrar, George S. Field, Samuel J. Fields, George S. Gatchell, R. F. Goodman, George Gorham, Charles A. Gould, John C. Graves, Edwin B. Guthrie, Richard Hammond, A. H. G. Hardwicke, Edmund Hayes, Dr. Leon F. Harvey, E. L. Hedstrom, R. R. Hefford, W. W. Hildreth, Thomas Hodgson, Lewis H. Knapp, J. N. Larned, George Howard Lewis, Thomas Loomis, J. J. McWilliams,

Capt. F. A. Mahan, U.S.A., George E. Mann, William Martin, Dunkirk; R. K. Noye, D. O'Day, William Pierce, Henry A. Richmond, George A. Ricker, Sherman S. Rogers, Benjamin Rhodes, Niagara Falls; N. S. Rosenau, B. C. Rumsey, F. A. Seabert, John Satterfield, L. G. Sellstedt, John Slote, John H. Smith, Pemberton Smith, F. H. Snell, George R. Sykes, James Tillinghast, Charles F. Wadsworth, Piffard; H. J. Warren, C. M. Whiting, Robert H. Williams, Julian J. Williams, Dunkirk.

Hotel-Headquarters, Niagara Hotel.

The first session was held on Tuesday evening, October 2d, at the rooms of the Buffalo Society of Natural Science, in the Buffalo Library Building. The meeting was called to order by Dr. Julius Pohlman, Secretary of the Local Committee, who delivered a graceful and cordial address of welcome.

Professor W. B. Potter, President of the Institute, after an appropriate acknowledgment of this reception, proceeded to deliver the Presidential Address. (This will be separately published.)

A curious gold breast-plate, found in excavations at a mine in the State of Panama, was exhibited by the Secretary.

At the second session, held at the Niagara Hotel, Wednesday evening, October 3d, the following papers were read:

Asphalt and its Uses, by F. V. Greene, New York City.

The Life-History of Niagara, by Dr. Julius Pohlman, Buffalo, N. Y.

Soaping Geysers, by R. W. Raymond, New York City.

At the third session, held at the Niagara Hotel, on Friday morning, October 5th, the following papers were read and discussed:

Steel Rails, and Specifications for their Manufacture, by Robert W. Hunt, Chicago, Ill.

The Present Status of Electrical Transmission of Power, by R. P. Rothwell, New York City.

The fourth and final session was held at the Niagara Hotel on Friday evening, October 5th, when the following papers were read by the Secretary in the absence of their authors:

The Mining Industry in Its Relation to Forestry, by B. E. Fernow, Washington, D. C.

Note on the Cultivation of Mushrooms in Abandoned Mines at Akron, N. Y., by W. Y. Warren, Buffalo, N. Y.

The following papers were read by title:

The Glenmore Iron Estate, Greenbrier Co., West Virginia, by W. N. Page, Powelton, W. Va.

Anthracite and Coke, Separate and Mixed, in the Warwick Blast-Furnace, by Edgar S. Cook, Pottstown, Pa.

Tuyere Slagging-Valve, by Edgar S. Cook, Pottstown, Pa.

The Chlorination of Low-Grade Auriferous Sulphides, by William B. Phillips, Chapel Hill, N. C.

A Differential Regenerative Hot-Blast Stove, and Its Application to an Open-Hearth Blast-Furnace, by Jacob T. Wainwright, Pittsburgh, Pa.

Note on a Specimen of Gilsonite from Uintah Co., Utah, by R. W. Raymond, New York City.

Notes on the Roasting of the Hudson River Carbonates, by Ingersoll Olmsted, Burden, N. Y.

The Minerals of Ontario and their Development, by W. Hamilton Merritt, Toronto, Canada.

Note on Cast-Steel Water-Jackets, by R. H. Terhune, Salt Lake City, Utah.

The Flue-Dust of the Furnaces at Low Moor, by Ellison C. Means, Low Moor, Va.

Notes on the Rosario Mine at San Juancito, Honduras, C. A., by Thomas H. Leggett, Fairplay, Colo.

The Impurities of Water, by A. E. Hunt and George H. Clapp, Pittsburgh, Pa.

Pig-Iron of Unusual Strength, by Fred. P. Dewey, Washington, D. C.

Note on the Nickel-Ore of Russell Springs, Logan Co., Kansas, by Fred. P. Dewey, Washington, D. C.

Cement-Rock and Gypsum-Deposits in Buffalo, by Julius Pohlman, Buffalo, N. Y.

The Geology of Buffalo, as Related to Natural Gas Explorations along the Niagara River, by Charles Albert Ashburner, Pittsburgh, Pa.

The Equalization of Load on Winding Engines by the Employment of Spiral Drums, by E. M. Rogers, Central City, Colo.

The Northwestern Colorado Coal-Region, by G. C. Hewett, Glenwood Springs, Colo.

The Effect of Velocity and Tension of Gases on the Reduction of Ores in the Blast-Furnace, by Theo. W. Robinson, Joliet, Ill.

Water-Gas as a Steam-Boiler Fuel, by D. S. Jacobus, Hoboken, N. J.

Ferro-Silicon and the Economy of its Use, by W. J. Keep, Detroit, Mich., and Edward Orton, Jr., Columbus, Ohio.

Hot Spring Formations in Red Mountain District, Colorado: A Reply to the Criticisms of Mr. Emmons, by Theodore B. Comstock, Champaign, Ill.

Large Furnaces on Alabama Material: Discussion Continued, by Fred. W. Gordon, Philadelphia, Pa.

The New Dressing-Works of the St. Joseph Lead Company, at Bonne Terre, Missouri, by H. S. Munroe, New York City. (Brought forward from the Boston meeting, February, 1888.)

The Secretary announced that the Fifty-Second (being the Annual) Meeting of the Institute would be held in New York City, beginning Tuesday, February 19th, 1889.

Notice was given of the following amendments to the Rules, proposed for adoption at the Annual Meeting:

It is proposed to amend:

Rule II., by striking out part or all of the final proviso.*

Also, by inserting after the words "by ballot at a regular meeting," in the seventh and eighth lines of the Rule, and also after the same words in the eleventh line of the Rule, the following parenthesis:

"(or by ballot at any time conducted through the mail, as the Council may prescribe)."

Rule V., by striking out the parenthesis, "(except honorary members)."*

Also, by inserting at the end of the first sentence (7th line of the Rule) the following words:

"and if the Council, or a Committee thereof appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper."

The following persons, having been duly recommended by the Council, were unanimously elected:

MEMBERS.

Walter T. Arms,	Punxsutawney, Pa.
Thomas S. Blair, Jr.,	Pittsburgh, Pa.
John Blue,	Capelton, P. Q., Canada.
Wilmot W. Burritt,	Englewood, N. J.
James L. Buskett, Jr.,	Elkhorn, Montana.
John H. Campbell,	Kingman, Arizona.
Thomas M. Chatard,	Washington, D. C.

* Previous notice of this amendment was given at the Boston meeting, February, 1888. See *Trans.*, XVI., xxxvi.

Clarence M. Clark,	Philadelphia, Pa.
Albert Conro,	Milwaukee, Wis.
T. Dan,	Chikugo, Japan.
John J. Deemer,	Chester, Pa.
A. J. Ehrob,	Pittsburgh, Pa.
Joseph S. Elverson,	Camden, N. J.
Percival Fowler,	London, England.
W. T. Goodnow,	Lebanon, Ky.
James G. Green,	Honda, Colombia, S. A.
F. V. Greene,	New York City.
Charles E. Hall,	St. Louis, Mo.
Jesse Hall,	Phœnixville, Pa.
J. A. Haskell,	Walston, Pa.
Alphonse Hennin,	Springfield, Ill.
Theo. M. Hopke,	Pittsburgh, Pa.
Asmus Jabs,	Bernburg, Germany.
W. Larimer Jones,	Pittsburgh, Pa.
W. J. Keep,	Detroit, Mich.
George D. Kelly,	Sharpsville, Pa.
James B. Knight,	Norway, Mich.
R. Kondo,	Tokio, Japan.
Horace W. Lash,	Pittsburgh, Pa.
Henry A. Lee,	Ouray, Colo.
Louis Mohr,	Chicago, Ill.
H. K. Myers,	Tyrone, Pa.
S. B. Nixon,	El Paso, Texas.
Ingersoll Olmsted,	Hudson, N. Y.
Francis T. Peacock,	Anniston, Ala.
James W. Pender,	Elkhorn, Montana.
Francis A. Pocock,	Boston, Mass.
George H. Potts,	Pottstown, Pa.
George E. Potts,	Walston, Pa.
F. H. Prentiss,	New York City.
I. M. Power,	London, Eng.
Camilo C. Restrepo,	Medellin, Colombia, S. A.
William H. Schlemm,	Durango, Mexico.
Francis M. Simonds,	New York City.
T. L. Skinner,	Terraville, Dakota.
Olaf Venström,	Marquette, Mich.
Fernando Sustersic,	Mexico.
Charles R. Trew,	Dunbar, Pa.
Dr. Leonard Waldo,	Lockport, N. Y.
William F. Wilkinson,	London, England.

ASSOCIATES.

George F. Kunz,	New York City.
Richard H. Morris, Jr.,	Bethlehem, Pa.
Augustus W. Newell,	Bradford, Pa.
E. C. Roberts,	Buffalo, N. Y.
C. D. Stowitz,	Buffalo, N. Y.
Willis B. Wilcox,	St. Louis, Mo.

The following associates were, upon recommendation of the Council, made members:

J. R. Bien,	New York City.
C. L. Burlingham,	Chicago, Ill.
W. A. Connell,	Portsmouth, Ohio.
Henry H. Porter, Jr.,	Mexico.
C. H. Smyth, Jr.,	Oneida Co., N. Y.
F. W. Sperr,	Mineral Park, Arizona.
George C. Stone,*	Newark, N. J.

After the passage of a resolution, instructing the Secretary to express by letter to the Local Committee and to various citizens, firms and corporations the thanks of the Institute for the cordial and abundant courtesies extended to its visiting members and guests, the meeting was adjourned.

Excursions and Entertainments.

On Tuesday evening, after the close of the first session, an informal reception was held in the Library Building, under the auspices of the Buffalo Library, the Society of Natural Science, the Academy of Fine Arts and the Historical Society.

Wednesday was occupied in an excursion by special train over the Western New York and Pennsylvania R.R. to Dunkirk, N. Y., where the Brooks Locomotive Works were visited, and the party was elegantly entertained by Mrs. Brooks, assisted by ladies of Dunkirk.

On Thursday, excursions were made to many points of interest in and around the city of Buffalo, including the Union Bridge Works, the grain-elevators, the coal- and ore-docks of the Lehigh Valley R.R. Co., the Government break-water, the water-works pumping-station, the works of the Buffalo Cement Co. (where gas-wells, quarries, kilns, etc., were inspected, and the party was entertained at luncheon), the Barber Asphalt Paving Co., the parks and asphalt-paved streets, the Wagner Palace Car Works, numerous machine-shops, etc.

On Thursday evening, a subscription-dinner took place at the Niagara Hotel.

On Friday, an excursion was made by special train to Lockport,

* To correct a clerical error committed in 1883, when Mr. Stone was made a member by vote of the Institute, and, the proper record being accidentally omitted, continued to be carried on the books of the Institute as Associate.

N. Y., where the works of the Holly Manufacturing Co. and the Cowles Electric Smelting and Aluminum Co. were visited, the party being conveyed to the latter in carriages.

On Saturday, an excursion was made by special train over the D. L. & W. R.R. and the W. N. Y. & Pa. R.R. to Piffard, N. Y., where the rock-salt mine of the Retsof Mining Co. was visited. Luncheon was served on the train.

MEMBERS AND ASSOCIATES PRESENT.

The following members and associates were registered at the headquarters of the Local Committee. The list is doubtless incomplete.

W. L. Austin.
Charles A. Ashburner.
James Archbald.
D. Brennen.
C. R. Claghorn.
Walter Crafts.
Edgar S. Cook.
P. Matson Clark.
W. B. Cogswell.
W. E. C. Coxe.
E. V. d'Invilliers.
E. M. Ferguson.
J. H. Gates.
Robert W. Hunt.
Alfred E. Hunt.
Samuel Hines.
R. W. Hildreth.
J. F. Holloway.
Henry M. Howe.
Thomas Hodgson.
L. Holbrook.
William Larimer Jones.
J. E. Johnson.
Asmus Jabs.
D. S. Jacobus.
C. Kirchhoff, Jr.
Alfred Markham.
P. N. Moore.
J. M. McLeary.
Edward Nichols.
William G. Neilson.
H. B. Nason.
William B. Potter.
S. Peters.
Francis A. Pocock.
George H. Potts.
J. Wesley Pullman.
J. A. Price.
Charles O. Parsons.
R. W. Raymond.
W. S. Russell.
T. W. Robinson.
R. P. Rothwell.
J. H. Sykes.
E. C. Schaufuss.
S. Taylor Sheaffer.
S. Stutz.
J. M. Sherrerd.
C. D. Simpson.
T. Guilford Smith.
C. E. Sutherland.
Francis M. Simonds.
Emanuel Trotz.
William Thaw, Jr.
N. B. Wittman.
Robert H. Williams.
E. B. Wilson.
William H. Wiley.
S. Whinery.
Rollin H. Wilbur.

PROCEEDINGS OF THE FIFTY-THIRD (NINETEENTH ANNUAL) MEETING, NEW YORK CITY, FEBRUARY, 1889.

LOCAL COMMITTEE.

Andrew Carnegie, *Chairman;* W. H. Wiley, *Secretary;* Wm. H. Adams, Geo. H. Babcock, O. W. Barnes, J. C. Bayles, H. W. Bulkley, Jas. A. Burden, Cyrus Butler, W. S. Church, E. F. De Camp, J. E. Denton, Thos. Egleston, F. A. Halsey, A. S. Hewitt, L. Holbrook, J. F. Holloway, W. R. King, Chas. Kirchhoff, Jr., J. F. Lewis, Chas. Macdonald, J. B. Mackintosh, S. F. Morris, H. S. Munroe, F. O. Norton, F. M Pierce, Fred. H. Prentiss, J. C. F. Randolph, R. W. Raymond, E. N. Riotte, Chas. M. Rolker, R. P. Rothwell, W. P. Shinn, E. G. Spilsbury, John Stanton, Grey Torrey, H. G. Torrey, W. W. Van Voorhis, John A. Walker, David Williams, F. S. Witherbee, E. L. Zalinski.

Hotel Headquarters.—Union Square Hotel.

The first session was held on Tuesday evening, February 19th, at Hardman Hall, Nineteenth street and Fifth avenue. The meeting was called to order by Mr. Andrew Carnegie, Chairman of the Local Committee, who cordially welcomed the Institute to the city. President Potter, after due acknowledgment of this greeting, proceeded to deliver the Presidential address. (This will be separately published.) The following papers were then read:

Biographical Notice of Erich C. Schaufuss, by J. H. Bowden, Wilkes-Barre, Pa.

End-Lines and Side-Lines in the U. S. Mining Law, by R. W. Raymond, New York City.

The second session was held in the library of the Edison laboratory, at Llewellyn, N. J., on Wednesday afternoon, February 20th, when the following papers were read:

The Concentration of Iron-Ore, by John Birkinbine, Philadelphia, Pa., and Thomas A. Edison, Llewellyn, N. J.

The Application of Electric Accumulators or Storage-Batteries in Mining, by P. G. Salom, Philadelphia, Pa.*

The Brush Electric Plant in the Comstock Mines, Virginia City, Nevada, by Nelson W. Perry, Cincinnati, O.*

*This paper has not been furnished by the author up to the closing of the present volume.

The third session was held at Hardman Hall, on Wednesday evening, February 20th, and was attended by a considerable number of the members of the American Society of Civil Engineers, the regular meeting of that Society on the same evening having been courteously suspended to permit a joint session with the Institute.

The President appointed Messrs. T. M. Drown, J. F. Lewis, and J. M. Sherrerd as Scrutineers to examine the ballots received for the annual election of officers, and report the result at a subsequent session.

The following papers were then read and discussed:

A communication from Mr. C. P. Sandberg, London, England, and a reply from R. W. Hunt, Chicago, Ill., in discussion of the paper of the latter on "Steel Rails," presented at the Buffalo meeting.

Rail-Sections, by Frederic A. Delano, Chicago, Ill.

Proposed Rail-Sections, by R. W. Hunt, Chicago, Ill.

The fourth session was held at Hardman Hall, on Thursday morning, February 21st. A large number of new members and associates were elected. (These names are included in the list below.) The following notes were read by Mr. Kirchhoff, in the absence of the authors:

Note on the Use of Crude Petroleum as Fuel for Raising Steam at the South Chicago Works, by E. C. Potter, Chicago, Ill.

Coal *versus* Oil in the Puddling-Furnace and in Raising Steam, by G. H. Billings, Boston, Mass.

Oil as a Metallurgical Fuel, by E. C. Felton, Steelton, Pa.

After a general discussion of the subjects of these notes, the following papers were read by their authors:

The Copper-Deposits of Copper Basin, Arizona, and Their Origin, by W. P. Blake, New Haven, Conn.

The Determination of Silicon in Ferro-Silicons; Its Occurrence in Aluminum as Graphitoidal Silicon; and a Study of its Reactions with Alkaline Carbonates, by H. J. Williams, St. Louis, Mo.

The fifth session was held at Hardman Hall, on Thursday afternoon, February 21st. The Scrutineers reported the following officers as elected:

PRESIDENT.

RICHARD PEARCE,	Argo, Colorado.

VICE-PRESIDENTS.
(To serve two years.)

ECKLEY B. COXE,	Drifton, Pa.
CHARLES MACDONALD,	New York City.
PERCIVAL ROBERTS, Jr.,	Philadelphia, Pa.

MANAGERS.
(To serve three years.)

J. H. BRAMWELL,	Roanoke, Va.
FRANK FIRMSTONE,	Easton, Pa.
W. H. PETTEE,	Ann Arbor, Mich.

SECRETARY.

R. W. RAYMOND,	New York City.

TREASURER.

THEODORE D. RAND,	Philadelphia, Pa.

The Secretary read the

ANNUAL REPORT OF THE COUNCIL.

In accordance with the rules, the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer, duly audited, shows receipts for the year from all sources, including the balance at the beginning of the year, of $24,937.47, and expenditures of $19,057.19, leaving a surplus of $5880.28. In addition to this, the Treasurer holds United States bonds belonging to the Institute as follows: 4 per cent. bonds, par value, $2900; 4½ per cent. bonds, par value, $4400; total, $7300, of which the present market-value is about $9000. The total amount of cash on hand and securities at par is $13,180.28, or, reckoning the securities at present market-value, nearly $15,000.

Since the balance from the last statement was $4615.15, the actual receipts of the year were $1265.13 greater than the expenditures.

The detailed statement is as follows:

Statement of the Secretary and Treasurer, of Receipts and Disbursements, from February 1st, 1888, to January 31st, 1889.

Balance from statement, February 1st, 1888,		$ 4,615 15
Received for dues from members and associates,	$15,972 21	
" " life-memberships,	200 00	
" " sale of publications,	1,848 90	
" " binding *Transactions*,	826 69	
" " authors' pamphlets,	722 59	
" " engraving and electrotyping,	325 64	
		$19,896 03
Interest on U. S. bonds and deposits,		426 29
		$24,937 47

Paid for printing Vol. XVI. of *Transactions*,			$2,583 40
" " "	pamphlet editions of papers,		1,881 35
" " "	authors' editions of papers,		717 08
" " "	index of vols. I.–XV. inclusive,		1,001 44
" " "	list of members,		123 50
" " "	mailing list,		17 00
" " "	circulars and ballots,		104 56
" " binding Vol. XVI. of *Transactions*,			925 77
" " "	other volumes of *Transactions*,		221 95
" " "	index of vols. I.–XV. inclusive,		389 94
" " "	exchanges,		100 49
" " engraving and electrotyping,			1,556 50
" " postage (including P.O. box rent),			1,089 98
" " incidental expenses of mailing books,			42 45
" " stationery (including postpaid envelopes),			460 25
" " rent of offices,			800 00
" " janitor's fees,			114 00
" " express charges, freight, etc.,			753 31
" " telegrams,			10 57
" " index of Vol. XVI. of *Transactions*, proof-reading,			100 00
" " insurance,			9 00
" " coal purchased,			23 00
" " small expenses unclassified,			79 52
" " salaries of secretary, assistants and stenographer,			5,410 00
" " expenses of Secretary at meetings,			402 40
" " map purchased for library,			10 00
" " storage on *Transactions* advanced for 1889,			129 73
			$19,057 19
Balance,			5,880 28
			$24,937 47

As special expenditure for the benefit of members, may be mentioned the publication and free distribution of the volume containing the index of Vols. I.–XV. of the *Transactions*. The cost of the preparation, proof-reading, etc., of this book was included in the annual statement of last year; that of type-setting, printing and delivery was paid this year. Altogether, the issue of this general index has involved an expenditure of about $2500. That it was not only a useful undertaking—perhaps the most useful to which that sum could have been devoted for the advantage of members—but also a wise business operation, is proved by the increased receipts from the sale of publications during the past year. The possession of such an index greatly enhances the convenience, and therefore the value, of the *Transactions* as a cyclopædia of modern knowledge and practice in the professions which they cover. Ordinary text-books can never be either so comprehensive or so detailed; and only

at intervals, and for short periods, can they adequately represent the most recent science or experience. On the other hand, the invaluable contributions to technical literature contained in the annual reports of professional societies, or in the scientific periodicals, are often only to be found by the student after much labor in searching through inadequate and scattered indexes.

Fortunately for the Institute, its income has been large enough to permit the preparation and distribution of an index, giving to the numerous volumes of its *Transactions* the convenient character of a single work. As was to be expected, a greater demand for the back volumes immediately made itself felt, resulting in a largely increased revenue from their sale. The amounts given in the annual statements as received from "sale of publications" include small sums paid for copies of single papers in pamphlet form. Neglecting these, as insignificant in proportion, the following statement will show how the receipts for back volumes have increased during the history of the Institute, since they became important enough to be separately mentioned in the annual reports of the Council:

Year ending February	Received from sale of publications.
1879,	$ 171 50
1880,	598 99
1881,	290 50
1882,	926 25
1883,	1102 00
1884,	559 75
1885,	740 67
1886,	785 33
1887,	472 86
1888,	1112 22
1889,	1848 90
	$8608 97

The stock of back volumes on hand has never been reported among the available assets of the Institute, although, as this statement shows, it has been a source of no inconsiderable revenue.

The volumes are not stereotyped; and the loss to the Institute would have been irreparable if the stock of the earlier ones on hand in 1879, had not been rescued by Dr. Drown from the conflagration which destroyed the building in which they had been deposited. With the exception of a small supply kept on hand at the Secretary's office, these books are now stored in a fire-proof warehouse.

Through the courage and foresight of the Council, which directed the publication of the first volume, an edition of 1500 copies was printed, although the number of members was so small at that time as to cause a financial deficiency, for which the individual members of the Council became liable. The rapid increase of membership, however, soon extinguished this debt, and vindicated the wisdom which had created it. The edition was subsequently increased (with Vol. XI.) to 2000 copies; then successively to 2100, 2200, 2300 and 2400, which will be the number printed of Vol. XVII., now in press. The result of postponing the first increase so long was that the supply of Vols. IX. and X. became disproportionately reduced.

The number of back volumes on hand at the present time is given in the following statement, together with their value at the price for which they are now sold, viz., Vols. I., II., III. and IV. at $2 each; V., VI., VII. and VIII. at $3 each; and the rest at $5 each, except vol. X., the price of which is $10. For bound copies (of which a few are kept on hand) $1 each is charged for the binding.

Volumes of Transactions on Hand.

Volume.	Number of Copies.	Value.
I.,	649	$1301 00
II.,	707	1416 00
III.,	634	1273 00
IV.,	518	1042 00
V.,	523	1575 00
VI.,	560	1687 00
VII.,	491	1478 00
VIII.,	453	1363 00
IX.,	331	1660 00
X.,	206	2066 00
XI.,	508	2545 00
XII.,	498	2495 00
XIII.,	509	2550 00
XIV.,	475	2383 00
XV.,	440	2213 00
XVI.,	555	2810 00
		$23,857 00

Three meetings have been held during the year: the eighteenth annual meeting (being the fiftieth of the Institute) at Boston, in February; a May meeting at Birmingham, Ala.; and an October meeting at Buffalo, N. Y. It need not be said that these meetings were both pleasant and instructive to those who attended them; and the papers for which they gave occasion do not fall below the standard which has been maintained in our *Transactions* hitherto.

It is an inevitable consequence of the present large membership of the Institute, distributed as it is throughout our own and other countries, that its benefits should be enjoyed by the overwhelming majority of its members chiefly or solely through its publications. Perhaps half the members have never attended a meeting. Certainly a much larger proportion have attended but two or three. Since the abundant revenue derived from this class is what permits the publication of our *Transactions* with such satisfactory fulness and care; and since the value of the *Transactions* is the main reason for the continuance of this revenue, and the main benefit bestowed by the Institute upon the arts and sciences which it represents, the maintenance of uniform excellence in this respect is matter for hearty congratulation thus far, and for solicitous effort in future. It is not too much to say that everything else should give way to this.

At the same time, the social and professional benefit of the meetings is intrinsically desirable, and constitutes a most important factor in the production of valuable papers for the *Transactions*. Judged by its principal operations, the Institute might seem to be no more than a great publishing and distributing agency; but it is, and should always continue to be, much more than that. The personal friendships and professional as well as social intercourse promoted by the meetings, and by the constant correspondence of members with one another, and with the Secretary's office, are benefits which a mere publishing agency could not confer.

It is earnestly desired that in every practicable way the interest of the meetings and discussions should be enhanced; and since most of the papers must necessarily be read by title, and few can be presented in print, opportunity will be given at each meeting for the discussion or continued discussion of any papers presented at former meetings, as well as for those first presented at that time. It is hoped that the practice of sending remarks by mail to the Secretary, in discussion of papers, will be extended more widely among the members. Such remarks, though published and circulated at first as separate sheets, can be in most cases subsequently appended to the paper to which they refer, on its final publication in the annual volume. Even when this is rendered impracticable by the circumstances, care will be taken to index them so that the reader of each volume may be guided to all the opinions, criticisms and statements of practice which it contains upon a given subject.

Changes in membership since the last annual report have been

as follows: 4 honorary members (of whom 2 were previously "foreign members,") 182 members and 20 associates have been elected and have accepted their election; 8 members have been reinstated; 14 associates have become members; 26 members and 7 associates have resigned; 34 members and 10 associates have been dropped from the roll (subject to reinstatement at the discretion of the council) for non-payment of dues; and the deaths of 19 members and 1 associate have been reported.

These changes are tabulated as follows:

	H. M.	F. M.	M.	A.	Totals.
At date of last report	7	47	1480	180	1714
Gains by Election	2	.....	182	20	204
" Change of Status	2	.....	15	.....	17
" Reinstatement	.....	.....	8	.....	8
Losses by Resignation	.....	.....	26	7	33
" Dropping	.....	.....	34	10	44
" Change of Status	.....	2	.....	14	16
" Deaths	.....	.....	19	1	20
Total gains	4	.....	205	20	229
" losses	.....	2	79	32	113
Present Membership	11	45	1606	168	1830

The list of deaths comprises the names of William Ashburner, D. H. Barrett, B. W. Cheever, S. M. Felton, Herbert C. Foote, Ignatius Hahn, Prof. R. D. Irving, Daniel N. Jones, G. Jones, E. Landsberg, A. H. Nicholson, Samuel Noble, Erich C. Schaufuss, R. H. Stodder, W. R. Symons, J. N. Tileman, H. A. Tuttle, T. Frank Walter, C. E. Wright, members, and W. H. Harvey, associate.

The exhibit of vigorous prosperity and continued increase in strength and usefulness constituted by the foregoing statistics needs no comment. It adds another to the long and unbroken series of similar reports presented to the Institute, and, with its predecessors, bears witness to the strong support invited, deserved and received by this society from the members of the profession it was organized to represent.

The following persons were (at this and the preceding session), upon recommendation of the Council, unanimously elected as members and associates respectively:

MEMBERS.

W. A. Akers,	Philipsburg, Mont.
James Allen,	Wellington, N. Z.
J. A. Allen,	Alliance, O.
Charles E. Andersson,	Denver, Colo.
Gustavus J. Bidtel,	Rising Fawn, Ga.
H. Birkinbine,	Council Bluffs, Iowa.
Harrington Blauvelt,	De Noon, Arizona.
Albert Broden,	Reading, Pa.
David H. Browne,	Ann Arbor, Mich.
Henry C. Carney,	Butte City, Mont.
John D Cheever,	New York City.
Spencer W. Clawson,	Bisbee, Arizona.
Henry S. Craven,	Boston, Mass.
Donald G. Crichton,	Montalbion, Queensland.
A. B. Davis,	Eureka, Nevada.
William E. Dickinson,	Bessemer, Mich.
P. W. Duffield,	London, England.
Thomas A. Edison,	Orange, N. J.
George H. Eldridge,	Washington, D. C.
Thomas Fisher,	Horatio, Pa.
Morcrieff Parry Gossett,	London, England.
Walter S. Greene,	Crown Point, N. Y.
Robert A. Hadfield,	Sheffield, England.
John S. Halsey,	Buena Vista, Colo.
Robert H. Hammond,	Philadelphia, Pa.
W. P. Headden,	Rapid City, Dakota.
Robert Henderson,	Glasgow, Scotland.
H. C. Holthoff,	Buena Vista, Colo.
Jacob W. Hoysradt,	Hudson, N. Y.
Millard Hunsiker,	Pittsburgh, Pa.
James T. B. Ives,	Toronto, Canada.
Elfric Drew Ingalls,	Ottawa, Canada.
Charles Davis Jameson,	Iowa City, Iowa.
Thomas G. Janvier,	Philadelphia, Pa.
Walter S. Kelley,	Helena, Mont.
Martin A. Lathrop,	Parral, Mexico.
Albert R. Ledoux,	New York City.
James G. Lindsley,	Rondout, N. Y.
William W. Lindsay,	Philadelphia, Pa.
J. M. McMahan,	New York City.
Herman C. Mannheim,	Pueblo, Colo.
P. A. L. Mannheim,	Pueblo, Colo.
Edward P. Mathewson,	Pueblo, Colo.
William S. Maxwell,	Brooklyn, N. Y.
Henry G. Merry,	Low Moor, Va.
John Da Costa Newbold,	Norristown, Pa.
A. W. Nibelius,	Hackettstown, N. J.
John D. Ormrod,	Emaus, Pa.
L. Orynski,	San Antonio, Texas.

Joseph Payette,	Ferrona, N. Y.
Otto Pfordte,	Lima, Peru.
O. B. Peck,	Chicago, Ill.
R. W. P. Richardson,	Scottdale, Pa.
Thomas Rickard, Jr.,	Eureka, Nevada.
Nathan H. Robertson,	Scranton, Pa.
Augustus M. Ryon,	Deer Lodge, Mont.
William E. Saunders,	Fayette, Mich.
Frederick Sharpless,	Houghton, Mich.
Morris R. Sherrerd,	Scranton, Pa.
Major A. J. Simmons,	Rapid City, Dakota.
Josef Simons,	New York City.
Edwin A. Sperry,	St. Elmo, Colo.
Francis L. Sperry,	Sudbury, Canada.
Henry M. Stanley,	Cuyahoga Falls, O.
Frank M. Stanton,	Atlantic Mine, Mich.
A. A. Stevenson,	Lewistown, Pa.
Robert Stevenson,	San Francisco, Cal.
H. R. Stoughton,	Shelby, Ala.
James B. Strawbridge,	Ivanhoe Furnace, Va.
Oscar Szontagh,	San Francisco, Cal.
A. O. Tate,	Orange, N. J.
Laurence W. Tatum,	Posa, Sonora, Mex.
George H. Thomson,	NewYork City.
Laurence E. Tierney,	Bramwell, W. Va.
Arthur Uddenberg,	Marquette, Mich.
Ferdinand Van Zandt,	New York City.
Ferdinand Von Leicht,	New Almaden, Cal.
M. E. Wadsworth,	Houghton, Mich.
George B. Wilkinson,	New York City.
Joseph L. Wills,	Buckingham, Quebec, Canada.
Ernest R. Woakes,	Honda, U. S. of Colombia, S. A.

ASSOCIATES.

R. G. Brown,	New York City.
H. N. Camp,	New York City.
Joseph L. Colby,	Milwaukee, Wis.
Justice Cox, Jr.,	Philadelphia, Pa.
Stuart Warren Cramer,	New York City.
Alfred de Castro,	New York City.
Percy LeRoy Fearn,	New York City.
L. W. Francis,	New York City.
Francis W. Hardwick,	London, England.
Fritz A. Heinze,	New York City.
Marmaduke B. Holt,	New York City.
Reginald F. Jopling,	New York City.
George W. Palmer,	Saltville, Va.
Ysidoro Ygnacio Polledo,	Cardenas R.R., Cuba.
Edwin H. Wedekind,	New York City.

CHANGE OF STATUS.

Thomas H. Hurlbert,	Duluth, Minn.
Charles F. Lacombe,	Denver, Colo.
Charles Murray,	St. Louis, Mo.
H. B. C. Nitze,	Carroll, Deer Lodge Co., Mont.
C. W. Pritchett,	Pine, Idaho.
N. C. Ray,	Butte City, Mont.
Rollin H. Wilbur,	South Bethlehem, Pa.

The following amendments to the Rules were discussed and adopted, subject to the approval of members by ballot, according to Rule VIII.:

RULE II.

Strike out the following words in the *Proviso:* "or to be members of the Council."

RULE V.

Strike out in the first sentence the words "stamped with the seal of the Institute;" and insert, at the end of the same sentence, the words "and if the Council, or a committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper."

The amended rule would read as follows:

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, together with a copy of this Rule, and the names of the persons ineligible for election to each office; and if the Council, or a committee thereof appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates, with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the person who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

Mr. E. C. Pechin offered a resolution, reading (after amendments suggested by Prof. T. Egleston and accepted by Mr. Pechin):

Resolved, That the annual meeting of the Institute be devoted to the transaction of such business as may be presented, and to the reading and discussion of papers, excursions being generally omitted.

After debate the resolution was laid on the table.

The following papers were read by title in the absence of their authors:

A Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis, by Clemens Jones, Hokendauqua, Pa.

The Wenström Magnetic Separator, by Robert A. Cook, New Brunswick, N. J.

Hollow Iron Pig-Patterns, by B. F. Fackenthal, Jr., Riegelsville, Pa.

Notes on the Electrolytic Assay of Copper, by William Glenn, Baltimore, Md.

Coal-Transfer of the Mt. Carbon Company, Limited, by W. N. Page, Powellton, W. Va.

Gold-Milling in the Black Hills, by H. O. Hofman, Rapid City, Dakota.

Note on the Koepe System of Winding from Shafts, by J. H. Harden, Phœnixville, Pa.

Ore-Deposits of the Black Hills of Dakota, by F. R. Carpenter, Rapid City, Dakota.

Note on the Influence of Columbite on the Tin Assay, by F. R. Carpenter and W. P. Headden, Rapid City, Dakota.

The Distribution of Phosphorus in the Ludington Mine, Iron Mountain, Michigan: a Study in Isochemic Lines, by David H. Browne, Ann Arbor, Mich.

A New System for Operating Regenerative Hot-Blast Stoves, by Jacob T. Wainwright, Pittsburgh, Pa.

Notes on the Bernice Anthracite Coal-Basin, Sullivan Co., Pa., by Clarence R. Claghorn, Birmingham, Ala.

Automatic Dumping Cradles for Mine-Cars, by H. S. Munroe, New York City.

The English *versus* the Continental System of Jigging: Is Close Sizing Advantageous? by H. S. Munroe, New York City.

Silicon in Cast-Iron, by W. J. Keep, Detroit, Mich., with Analytical Determinations by H. S. Fleming, Emporium, Pa., and Edward Orton, Jr., Columbus, O.

The Occurrence and Treatment of the Argentiferous Manganese-Ores of Tombstone, Arizona, by Charles W. Goodale, Butte City, Montana. (Originally presented at the Utah and Montana Meeting of 1887, but not printed at that time.)

A Note upon a Modification of the Reducing Process used by the Carbon Iron Company, by A. E. Hunt, Pittsburgh, Pa.

Soaping Geysers, by Arnold Hague, Washington, D. C.

A Review of the Iron-Mining Industry of New York for the Past Decade, by John C. Smock, Albany, N. Y.

A Rapid Method for the Determination of Phosphorus in Certain Ores, by T. Reed Woodbridge, Port Henry, N. Y.

Supplementary Note on Blast-Furnace Lines, by Edward Walsh, Jr., St. Louis, Mo.

Prominent Sources of Iron-Ore Supply, by John Birkinbine, Philadelphia, Pa. (Originally announced at the Birmingham Meeting.)

The sixth and concluding session was held at the Stevens Institute of Technology, Hoboken, N. J., on Friday morning, February 22d, when the following papers were read and discussed:

Note upon Some Results of the Storage of Water in Arizona, by W. P. Blake, New Haven, Conn.

Nails from Tin-Scrap, by Oberlin Smith, Bridgeton, N. J.

The Reopening of the Tilly Foster Iron-Mine, by F. H. McDowell.

The Economy of Modern Air-Compressors, by J. E. Denton, Hoboken, N. J.*

Excursions and Entertainments.

With the exception of a brief visit to the Spiral Weld Tube Works at East Orange, on the way to the session of Wednesday, at Llewellyn, N. J., there were no excursions during the meeting. After the close of the last of the six working-sessions (the largest number ever held during a single meeting of the Institute), various points of interest in New York City were visited on Friday afternoon by such members, individually or in parties, as chose to make such use of the cordial invitations extended to them. On Saturday morning a large number of members and guests, who had remained in the city after the conclusion of the meeting, enjoyed an excursion by

*This paper has not been furnished in time for publication in the present volume.

steamer to the shops of the Central Forge Co., Whitestone, L. I., the U. S. Engineer Post and Torpedo Station, Willets Point, L. I., and the Zalinski dynamite gun at Fort Hamilton. Luncheon was furnished on board the steamer by the Local Committee.

Mr. Thomas A. Edison, before the session of Wednesday afternoon, and President Henry Morton, of the Stevens Institute, after the closing session of Friday morning, entertained the members and guests at luncheon. On Friday evening a brilliant social reception was given in honor of the Institute, by Mr. and Mrs. A. S. Hewitt.

On Thursday evening a subscription dinner took place at the Brunswick Hotel.

Members, Associates and Guests Present.

The following names were registered at hotel headquarters:

W. H. Adams.
A. W. Allen.
James Archbald.
J B. Austin.
W. L. Austin.
Charles Bailey.
James C. Bayles.
F. W. Barnes.
O. W. Barnes.
Alfred L. Beebe.
Dr. Robert Bell.
G. H. Billings.
Theodore A. Blake.
William P. Blake.
H. M. Boies.
J. H. Bowden.
Jerome L. Boyer.
J. H. Bramwell.
Albert Broden.
George Brooks.
Henry W. Bulkley.
M. D. Burke.
J. P. Carson.
S. H Chauvenet.
Walter S. Church.
C. R. Claghorn.
F. L. Clerc.
F. Collingwood.
H. V. Conrad.
C. Constable.
James Constable.
G. G. Converse.
Edgar S. Cook.
Torbert Coryell.
A. H. Cowles.
W. E. C. Coxe.
G. R. Cullingworth.
H. M. Curry.
C. W. Davenport.
David T. Day.
A. H. DeCamp.
W. S. DeCamp.
E. V. d'Invilliers.
J E. Denton.
Frederick P. Dewey.
W. F. Downs.
T. M. Drown.
P. H. Dudley.
A. Faber Du Faur.
W. F. Durfee.
E. D. Edgerton.
T. Egleston.
Karl Eilers.
C. E. Emery.
A. H. Emery.
S. F. Emmons.
William E. C. Eustis.
J. D. Evans.
J. S. Fackenthal.
Clark Fisher.
Thomas Fisher.
R. G. Ford.
Homer T. Fuller.
Robert S. Fulton.

H. D. Gamble.
Albin Garrett.
S. D. Gifford.
John L. Gill, Jr.
C. H. Glasser.
William Glenn.
G. M. H. Good.
Frederick W. Gordon.
Walter Graham.
George Ross Green.
D. F. Haasis.
William Hainsworth.
Edward B. Harden.
John H. Harden.
G. C. Hewett.
George Henry Hewitt.
Robert Henderson.
Samuel Hines.
L. Holbrook.
J. F. Holloway.
Marmaduke Holt.
H. M. Howe.
W. S. Hungerford.
Alfred E. Hunt.
R. W. Hunt.
A. O. Ihlseng.
James T. B. Ives.
Thomas G. Janvier.
C. H. Jouet.
Henry D. Kerr.
Joseph C. Kent.
William Kent.
Charles Kirchhoff, Jr.
George F. Kunz.
E. K. Landis.
Robert H. Lamborn.
H. W. Lash.
L. G. Laureau.
E. D. Leavitt, Jr.
R. G. Leckie.
A. E. Lehman.
George M. Lehman.
E. A. Lewald.
J. F. Lewis.
Arthur McClellan.
Andrew S. McCreath.
F. H. McDowell.
J. W. McGenniss.
R. C. McKinney.
J. M. McMahan.
Charles Macdonald.
J. B. Mackintosh.
Arthur S. Mahoney.
Alfred Markham.
Charles A. Marshall.
R. L. Martin.
Charles C. Mattes.
G. W. Maynard.
C. A. Meissner.
F. J. H. Merrill.
P. N. Moore.
Henry G. Morris.
S. Fisher Morris.
H. S. Munroe.
George C. Munson.
H. B. Nason.
William G. Neilson.
Augustus W. Newell.
George Ormrod.
John D. Ormrod.
Charles O. Parsons.
C. Q. Payne.
F. N. Pease.
E. C. Pechin.
Nelson W. Perry.
W. H. Pettee.
W. F. Pinkham.
F. E. Platt.
R. E. Plumb.
Francis A. Pocock.
J. N. Pott.
William B. Potter.
George H. Potts.
Frederick H. Prentiss.
J. A. Price.
Theodore D. Rand.
J. C. F. Randolph.
R. M. Raymond.
R. W. Raymond.
W. H. Rea.
George S. Rice.
R. H. Richards.
P. de P. Ricketts.
Charles Ridgely.
George W. Riggs.
E. N. Riotte.
F. C. Roberts.
C. Snelling Robinson.
D. S. Robinson.
S. J. Robinson.
Charles M. Rolker.
C. Henry Roney.

John E. Rothwell.
R. P. Rothwell.
P. G. Salom.
H. J. Seaman.
E. H. Sears.
W. L. Scaife.
F. Z. Schellenberg.
E. J. Schmitz.
W. H. Scranton.
A. W. Sheafer.
A. H. Sherrerd.
J. M. Sherrerd.
J. M. Silliman.
J. C. Simpson.
F. M. Simonds.
Frederick J. Slade.
Oberlin Smith.
J. C. Smock.
E. Gybbon Spilsbury.
Albert Spies.
George C. Stone.
Charles Taylor.
W. J. Taylor.
H. G. Torrey.
D. Townsend.
J. W. Tyson.
Edward A. Uehling.
Leonard Waldo.
John A. Walker.
Bard Wells.
Maunsel White.
William White, Jr.
Jerome Wheelock.
John F. Wilcox.
W. H. Wiley.
George B. Wilkinson.
David Williams.
Henry J. Williams.
J. Francis Williams.
E. B. Wilson.
F. S. Witherbee.
N. B. Wittman.
Howard Wood.
Francis Wyatt.
James E. York.

PAPERS.

THE LOSSES IN ROASTING GOLD-ORES AND THE VOLATILITY OF GOLD.

BY SAMUEL B. CHRISTY, PROFESSOR OF MINING AND METALLURGY, UNIVERSITY OF CALIFORNIA, BERKELEY, CAL.

(Birmingham Meeting, May, 1888.)

My attention was first definitely called to this subject in 1880 by Mr. Chas. H. Crosby, then in charge of the Pioneer Reduction Works at Nevada City, California. He stated that he had found extremely great difficulty in treating the sulphurets from the Murchie Mine. This mine was at that time producing considerable quantities of concentrated sulphurets that often assayed as high as $150 per ton. About three-fourths of this value was in gold, the rest in silver. In order to extract the silver content it was necessary to roast with salt. Without salt Mr. Crosby said he could roast the ore with very little loss of either metal, but the moment salt was added losses began to show themselves very rapidly. Thus, according to his tests, with 3 per cent. salt, the gold loss was 30 per cent. and the silver 50 per cent. of the assay value. He attributed these losses to tellurides which he supposed to be present. Through his kindness I secured a quantity of this ore, and with it I have made a large number of tests, with the assistance of several of my students, at the Mining Laboratories of the University of California. Some of the more important results of this work I shall present in this paper.

I have also had occasion to examine into the literature of the subject, and, curiously enough, in spite of its great practical importance, there seems to be very little definite information in print concerning it. Many of our standard authorities are either silent on the question or contain erroneous statements. The best information is in a shape hardly accessible to those who need it, and consequently I have thought it advisable to include a brief abstract of what I have been able to glean, in a shape for convenient reference.

The Effect of Heat Alone on Metallic Gold.

Gmelin and Kraut, in their *Handbuch der Chemie*, Bd. III., p. 1065, edition of 1875, say: "In the highest furnace heat, in the

burning mirror, and in the oxyhydrogen flame, gold is only volatile to a small extent. A silver plate held over gold which is strongly heated in the focus of a burning mirror is slowly gilded (Homberg). In melting platinum which contains gold, the latter is volatilized and may be collected by condensing it (Deville and Debray). Compare Fuchs on volatilization of gold in roasting ores containing volatile metals (Fuchs, *Wien Akad.*, Ber., Oct., 1850); by melting by itself or in alloys (Napier). Pure gold placed on the surface of porcelain in a good furnace fire, according to Elsener, volatilizes in part, but not entirely."

I have been unable to find the reference to the work of Fuchs, but an abstract of it is given in the *Engineering and Mining Journal*, August 22d, 1885, together with a curious list of many of the older authorities on the subject, by Dr. Henry Wurtz. According to Dr. Wurtz, Fuchs roasted black copper, and found losses due, in his judgment, to the presence of the volatile metals—arsenic and antimony, and not to sulphur, sulphides, or metallic oxides. After roasting the black copper till losses ceased, he treated the ore with sulphuric acid and roasted again. He found losses of gold and silver so long as coal (*charcoal?*) was present in the mass. Homberg, in 1709, claims to have found appreciable quantities of gold in pyrites that showed no traces of it after being roasted. I shall come to this subject later on.

Fehling's *Handwörterbuch der Chemie*, Bd. III., p. 476, states that the discharge of electricity of high tension from fine gold points volatilizes gold.

In addition to the above I would add that gold is easily volatile at the point of a good mouth blow-pipe flame. It is necessary to take a small fragment of gold, not larger than a pin-head. If this is melted on a Plattner cupel with the outer tip of the oxidizing flame the outer margin of the bone-ash soon becomes colored purple with volatilized gold.

The Effect of an Oxidizing Roasting on Gold Ores.

Of course, such temperatures as these never occur in roasting gold ores, and it does not follow because gold is volatile at a melting temperature that it is volatile in oxidizing roasting.

Plattner, in his *Metallurgische Röstprozesse* (Freiberg, 1856), goes very minutely into the losses of gold and silver in oxidizing roasting. He shows by a long series of tests made on the small scale in the muffle that a loss of silver in oxidizing roasting is unavoidable.

Thus, in some fifty tests, varying from ¼ of an hour to 1½ hours in duration, he records losses of from 0.5 per cent. to 18 per cent. He concludes (p. 126) that the percentage loss of silver increases with the temperature of roasting, with the looseness or porosity of the roasting charge, that is, with the facility with which the air can come into contact with the silver, and the freedom of the silver from combination with other substances. The loss also increases with the time of roasting. He also concludes that the silver is volatilized in the condition of oxide which decomposes at a lower temperature into silver and oxygen.*

With regard to gold contained in metallic sulphides statements had been made prior to Plattner's time that considerable losses had taken place when these were subjected to an oxidizing roasting, amounting in some cases to 100 per cent. (vide Homberg, previously cited). He, therefore, made a large number of roasting tests, both direct and synthetic. In these last he used artificial mixtures containing known amounts of gold, either as fine dust or as artificial sulphides and arsenides. He concludes (pp. 127–129) that a loss of gold can take place in an oxidizing roasting only when the operation is carried on so rapidly that fine particles are carried off mechanically by the draft. He also states that the celebrated Winkler had come independently to the same conclusion.

With regard to oxidizing roasting Küstel, in his *Roasting of Gold and Silver Ores*, 1880, says, p. 58: "Generally no loss of gold is suffered during the roasting, neither with iron pyrites, nor with arsenical pyrites, although exposed to a considerable heat and for a long time (from 24 to 48 hours). Many manipulators, however, who have had much experience in roasting gold sulphurets, complain of considerable loss of gold with some kind of sulphurets, which they could not avoid in spite of all experiments with reference to heat and general treatment. It is, therefore, always advisable to investigate the loss of gold with new sulphurets by assay."

On p. 57 Küstel records the loss of 20 per cent. of the gold-content in the oxidizing roasting of certain tellurides of gold and silver, and states that it is not a mechanical loss, but is due to volatilization. That this is so with certain tellurides of gold is undoubtedly true, though I have not yet had sufficient opportunity to thoroughly investigate this branch of the subject.

* Prof. Percy, *Silver and Gold*, vol. i., p. 18, throws some doubt on this last conclusion, and further experiments are perhaps necessary to settle the matter. As to the amount of the loss there is no doubt.

Experiments with the Oxidizing-Roasting of Murchie Pyrite.

In order to thoroughly test the conclusions of Plattner, as far as the Nevada City ores were concerned, I have made in my laboratory at various times a number of tests. The sulphurets were carefully examined, and were found to contain chiefly pyrite, with small amounts of chalcopyrite (about 0.5 per cent. to 1.5 per cent. of metallic copper), a little galena, very small amounts of quartz, and traces of quicksilver from the batteries. Analysis showed, besides, reactions for arsenic and antimony in traces, but although carefully tested for, no reaction for tellurium could be obtained. I understand, however, that in some samples of the ore from this mine traces of selenium and tellurium have been reported.

In 1881 Mr. Douglas Lindley, one of my students, made in duplicate two careful oxidizing roasts with this ore. The charges were 2 assay tons each. They were roasted in the muffle at a low red heat for an hour, and the temperature was raised to a full red-heat for half an hour more. In 1882 I roasted two charges of 1 assay ton (29.166 grammes) each of this same ore. They were roasted for one hour at a very dull-red heat with the covers on the roasting-dishes; then for another hour at the same temperature with the covers off, and were finally finished at a cherry-red for half an hour. Total time, 2½ hours. In 1886 Mr. Franklin Booth, my assistant, made another roast with this ore. Charges were 1 assay ton each. Temperature was at first incipient red for half an hour, then dull-red for 8 hours, the total time being 8½ hours.

In each of these cases the lot of ore was very carefully sampled, and the raw ore assayed by scorification; the whole of the roasted ore was then assayed by the crucible assay. The results were as follows:

TABLE I.

Oxidizing Muffle-Roasts.

	Raw Ore. Oz. per ton.		Roasted Ore. Oz. per ton.		Percentage Loss. Per ton.	
	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
1881, Lindley, 1½ hours, .	4.58	27.50	4.58	26.44	0.00	3.85
1882, Christy, 2½ " .	4.58	27.65	4.58	27.07	0.00	2.09
1886, Booth, 8½ " .	4.50	28.39	4.50	27.39	0.00	3.52

In order to obtain the above results, however, the most minute precautions had to be taken to avoid all causes of mechanical loss.

Roasting-covers were used till all danger of decrepitation was over, and stirring was avoided, as well as all excess of draft. The samples were exposed in a thin layer, and the roasting was in all cases very complete. The balance used was sensitive to less than 0.01 mg. Hence in the 1 assay ton lots any loss of gold above 20 cents to the ton would have been shown by the balance, and with the 2 assay ton lots above 10 cents per ton would have been shown. This seems to explain Homberg's statement and that of others that gold could not be found in pyrite after oxidizing roasting, although traces could be found before roasting. The only conclusion seems to be that such roastings were so conducted as to cause a mechanical loss by dusting.

Hence we may fairly conclude that Plattner's statements concerning the non-volatility of gold during the conditions that maintain in the oxidizing roasting of auriferous pyrite are entirely correct. And it may be fairly assumed that where a loss of gold is reported in such cases that it is due to a mechanical loss due to excess of draft or to insufficient or improperly arranged dust-chambers.

The Relations of Gold to Chlorine.

The Plattner chlorination process has made familiar to almost every one the fact that when gold is exposed to the action of chlorine gas in the cold, it is attacked and the soluble chloride of gold, $AuCl_3$, is formed. According to most authorities there exist two other chlorides of gold; the subchloride AuCl, formed when $AuCl_3$ is heated to 130° to 200° C. and a double chloride AuCl, $AuCl_3$ formed by warming AuCl or gold sponge in $AuCl_3$. Pratt states that when $AuCl_3$ is heated in chlorine a higher chloride than $AuCl_3$ forms.

In Gmelin-Kraut I find the statement (l. c. p. 1016) "molten gold is not attacked by chlorine." There is no alleged basis for this statement, but a simple reference to the Miller chlorination process for refining gold, where no further explanation is given. This statement is, as I shall show later, entirely wrong. It seems strange to find so misleading a statement in such an authority.

Prof. Percy (*Silver and Gold*, Part I. p. 402) gives an account of experiments made in 1838 by Messrs. Lewis Thompson and Arthur Aiken for "a method of assaying and purifying gold." Mr. Thompson received a prize of 20 guineas for his method from the Society of Arts. Mr. Thompson says of his method that it is "founded

upon a circumstance long known to chemists, viz.: that not only has gold no affinity for chlorine at a red heat, but that it actually parts with it at that temperature, although previously combined; that is to say, the chloride of gold is reduced to the metallic state by heat alone (and), it cannot, therefore, possess any affinity for chlorine when red hot; this, however, is not the case with those metals with which gold is usually alloyed; it offers, therefore, at once an easy and certain means of separation."

A further account shows, however, that they did not confine themselves to passing chlorine over gold at a red heat, but also used a melting heat; in one case the tube is spoken of as white hot. These experimenters state that, taking care to observe certain precautions that they mention, the gold was refined without any loss being perceptible on a scale that indicated to within 0.01 to 0.005 grains (0.6 to 0.3 milligrammes). I also note the mention of "a little chalk or common salt" with the gold although no reason for its presence is given except that Mr. Aiken states that "the presence of the alkaline chlorides seems to have the effect of preventing the volatilization of the chloride of silver" formed in refining.

I come next to a statement in direct opposition to that of the previous writers:

In the *Chemical News* of March 12th, 1869, appears a note (anonymous) on "The Volatility of Gold at a Cherry-Red Heat in a Current of Chlorine Gas." The writer recommends this process for the assay of gold quartz. He says: "If the pulverized quartz is placed in an earthenware tube of fireclay, and the same is heated in a furnace till the interior of the tube is a bright cherry-red, then if under these circumstances a current of chlorine gas be passed through the retort, the gold combined in the rock will combine at the high temperature with the chlorine, and become volatile therewith, whereas at the place where the heat of the tube or retort is less high the chloride of gold will be again decomposed and gold deposited."

Finally in the *Comptes rendus* of November, 1869, Debray makes a contribution to the subject by stating that while gold chloride is disassociated at 200° C. by heat alone that it is volatile at 300° C. in an atmosphere of chlorine. He also says that Boyle was the first writer who had stated that gold chloride was volatile to some extent without dissociation.

The above would appear to be the present conflicting and unsatisfactory state of knowledge on this subject. Let us next see what has been published on the losses of gold in chloridizing roasting.

The Losses of Gold in Chloridizing-Roasting.

Plattner does not seem to have been aware of the volatility of gold in chloridizing-roasting; for nowhere in his *Metallurgische Röstprozesse* does he make any reference to the fact, probably because in his time gold ores were subjected to an oxidizing roasting only, in preparing them for the Plattner chlorination process. In speaking of the effect of a chloridizing-roasting on the different metals he says on p. 271, after speaking of the volatilization of silver chloride when it comes in contact with other easily volatilized chlorides, "so that in chloridizing-roasting, under certain circumstances a not inconsiderable loss of silver may take place."

"Gold (in the native state, as well as when combined with other metals) is only when in a fine state of subdivision and in a slightly warmed state converted into the chloride, $AuCl_3$, which as already stated, gives up 2 atoms of Cl while still below a red heat and changes into AuCl, and this is by further heating transformed into metallic gold." He then goes on to state the necessity of passing chlorine gas through the previously roasted ore in order to chloridize the gold. But though he treats at length of the volatile products of chloridizing-roasting, he fails to make any mention of the losses of gold in this connection.

The only mention of the subject I have been able to find in Küstel is on p. 57, where, in speaking of the tellurides of gold, he says: "If salt is present during the roasting, the chloride of tellurium volatilizes voluminously, and it is possible that under this condition the tellurium causes the gold to volatilize likewise. The author, by his own experience—having 4 per cent. of salt in the ore on account of the silver—found the loss of gold amounting to 8 per cent. before the ore was properly red hot. This was two hours after charging. . . . . Whether the above loss would have occurred in such proportion if no salt had been added, cannot be stated."

Apparently, the first writer who published anything definite on the losses of gold in roasting with salt was Mr. C. H. Aaron. In his *Leaching Gold and Silver Ores* (1881), he says, p. 121, that he was first painfully made aware of the fact of this loss by having to make good a deficiency of some $3000 below the yield he had guaranteed. The ore was simple pyrite with no visible peculiarity. It was roasted in a three-hearth reverberatory, with 1 to 2 per cent. of salt added on account of the silver. He states that he was first led to suspect the cause of the loss from the examination

of a yellow sublimate that formed on the masonry of the furnace over the working doors, when the draft was reduced. This sublimate he "found to be very rich in gold, although there was none to be seen in it by the most careful washing. It also contained iron perchloride, and copper chloride, with lead and other substances."

He then instituted tests on the small scale with two similar samples, one with 4 per cent. of salt, the other without, the "roasting being pushed purposely to an extreme as to heat and time, and when the two tests were assayed under exactly similar conditions, that which was salted was found to contain less than half as much gold as the unsalted one."

He further adds: "I then took some light fluffy sublimate from the flue of a roasting furnace, an assay of which gave me a value of some $600 per ton, chiefly gold. The quantity of this material was, however, very small, and the bulk of the matter in the dust chamber was not richer than average bulk of the ore treated, *a circumstance which indicates that the gold was actually to a great extent volatilized in some not easily condensible form.*" The italics are mine.

Mr. Aaron further adds: "I also found that the ore sustained a loss of weight in roasting, equal to about 18 per cent., consequently the roasted ore ought to have been more than 18 per cent. richer than before roasting, which was not the case. If this is not considered sufficient proof that the gold may be volatilized in the roasting of some ores with salt, the deficiency is supplied by the fact that, as soon as I made the necessary change by reserving the salt until the nearly dead roasting of the ore was finished, not only did the roasted ore assay 20 per cent. richer than the raw, but the yield overran my guarantee, while the tailings nevertheless contained considerably more gold than before. . . . . I afterwards found that a very small quantity of salt, not more than three pounds to the ton, might be mixed with the crude ore without detriment to the gold, and with decided advantage to the extraction of the silver."

Mr. C. A. Stetefeldt gives (*Trans.*, xiv., 339) the very interesting results of his investigations on the chloridizing roasting of the gold ore from Las Minas, in the State of Vera Cruz, Mexico. The ore he treated consisted mainly of 43 to 67 per cent. of magnetite, 3 to 22 per cent. of pyrite and 3.5 to 7 per cent. of chalcopyrite, the remaining minerals being quartz and garnet. The ore contained less than one ounce per ton of gold and traces only of silver. He found that the losses in a chloridizing roasting were from 42.8 to 93 per cent. of the total gold content. He further states "there is no

doubt that the volatilization of the gold takes place with that of the copper chlorides. The loss increased with the quantity of these chlorides formed and volatilized." He further states, however, "That the copper chloride is by no means an essential element in producing this loss is shown in the following experiments, made by Mr. C. Butters with a gold ore entirely free from copper." This ore was a hard white quartz intimately mixed with about 7 per cent. of calcite and a little pyrite. It contained 5.55 oz. silver and 0.65 oz. of gold per ton. On subjecting this ore to an oxidizing roasting a loss of 2 to 9 per cent. of silver took place but no loss of gold. But when the same ore was subjected for one hour to a chloridizing roasting in the muffle at a cherry-red heat with 5 per cent. of salt, a loss resulted of 70 to 80 per cent. of the silver and 68 to 85 per cent. of the gold. On increasing the salt to 10 per cent. no increase of the losses took place.

I ought also to call attention to the claim of Mr. Stetefeldt that the loss of gold is less in the instantaneous roasting of the Stetefeldt furnace than in the more prolonged treatment in the reverberatory furnace. This claim does not appear to be unreasonable; and the indirect proof that he cites from the work of his furnace at the Lexington Mill on an auriferous silver ore tends to bear out that claim (see his paper, l. c., p. 343). This claim, if it could be established beyond doubt, would be of such great importance in the treatment of a large class of auriferous silver ores that must be roasted with salt on account of their silver content that I hope that he will find it possible to make sufficient direct tests on the large scale with the Stetefeldt furnace to settle the question.

In 1882, while engaged in the study of this subject, I sent Mr. Charles E. Hayes, then one of my students, to investigate what was at that time considered one of the best chlorination-works in California. I secured permission from the proprietors for him to make a thorough study of the work that they were actually doing, and they gave him every opportunity to make the study. They have also kindly allowed me make public the results, omitting mention of their names.

The furnace used was a double-hearth reverberatory, one hearth being placed above the other. The hearths were 18 feet by 15 feet. The concentrated sulphurets were first dried on the furnace-top for 36 hours. From here a charge of 2500 lbs. was dropped on to the upper hearth. Here it was roasted for 12 hours, when a second similar charge was put in. At the end of a second 12 hours the

first charge, which had now been in the furnace 24 hours, was dropped on to the lower or finishing-hearth, and a new charge put in the furnace to take its place. After remaining on the finishing-hearth about 8 hours, till the sulphurous fumes had ceased, 1 per cent. of salt was added, and 4 hours later the charge was drawn. This made a total time in the furnace of 36 hours.

Mr. Hayes found it impossible to keep the various charges in the furnace sufficiently distinct to follow any one of them through, so, with the consent of the proprietors, a single charge was put through the furnace by their workmen, under as nearly the usual conditions, as to temperature, etc., as possible. A charge of 2441 lbs., which had been dried as usual on the furnace-top for 36 hours, was weighed out and carefully sampled by Mr. Hayes.

The notes of the test were as follows:

March 26, 8 A.M. Charge of 2441 lbs. put on upper hearth.

March 26, 3 P.M. (after 7 hours). Charge dropped to the lower hearth, so as to cause as little delay in the use of the upper hearth as possible.

March 27, 11 A.M. (20 hours later). 22 lbs. of salt were added.

March 27, 5.30 P.M. (6 hours, 30 minutes). Charge was drawn. Total time, 33 hours 30 minutes.

The roasted ore weighed only 1873 lbs., having lost 23.27 per cent. of its weight in the roasting. The roasted ore was carefully sampled by Mr. Hayes, and the samples were assayed as follows:

TABLE II.

	Gold.	Silver.
Raw ore (average of 5 scorification-assays)...	4.536 oz.	19.884 oz. per ton.
Roasted ore (average of 4 crucible-assays)...	2.980 "	18.583 " " "
Hence $\frac{2441}{2000}$ tons raw ore contained.....	5.536 "	24.268 oz. Troy Net.
And $\frac{1873}{2000}$ tons roasted ore contained.....	2.791 "	17.403 " " "
Hence the actual net loss in roasting was.....	2.745 oz.	6.865 oz. Troy.
Or in per cent. of original ore-content.....	49.58% Gold.	28.28% Silver.

This enormous loss was entirely unexpected by the proprietors, who had thought it unnecessary to carry on any systematic sampling and assaying. It is but fair to add, however, that the mode of roasting in this test was not quite the same as the usual one, as the charge was on the lower hearth 26½ hours instead of 12 hours, and the salt was mixed with the ore 6½ hours instead of 4 hours, the usual time. Still the test shows in the most unmistakable manner

what enormous losses may take place in roasting such ores with salt, unless the most minute details of the treatment are carefully looked to. At these works the upper hearth was usually kept at a red heat a good part of the time, and the temperature of roasting was undoubtedly kept generally too high.

Mr. Hayes also sampled and assayed the tailings after they had been leached for gold and silver in the usual way with the following result:

	Gold. Ozs. per ton.	Silver. Ozs. per ton.
Average of 3 crucible assays of tailings, .	1.487	14.233

He, however, neglected to determine the weight of the tailings after leachings. If, however, we allow a loss of 10 per cent. in weight due to soluble salts, which is certainly fair to the process, we shall have as the net content of the tailings:

	Gold. Ozs. per ton.	Silver. Ozs. per ton.
$\frac{1117}{1000}$ tons tailings contain net, . . .	1.254	12.005

Or in percentage of original content there were lost in the tailings 22.65 per cent. of the gold and 49.47 per cent. of the silver.

Hence the final results of the test were:

TABLE III.

	Gold.		Silver.	
Loss by volatilization and dusting.....	2.745 oz.	or 49.58%	6.865 oz.	or 28.28%
Loss in tailings.....	1.254 "	" 22.65%	12.005 "	" 49.47%
Actual yield by mode of treatment.....	1.537 "	" 27.77%	5.398 "	" 22.25%
Original content raw sulphurets.....	5.536 oz.	or 100%	24.268 oz.	or 100%

A curious fact is that in spite of the long chloridizing roasting, perhaps in consequence of it,* less than 23 per cent. of the silver was extracted by the hyposulphite solution, and nearly 50 per cent. was left in the tailings. The excessive losses were in part due, no doubt, to the fact that this furnace had only recently been erected at the time of this test, and the workmen had hardly got used to it. For in 1880, Mr. H. W. Carroll, another of my students, obtained the following results from sampling the ore from the same mine and the final tailings. The ore had been roasted in an old-fashioned, long, two-step reverberatory furnace:

* As claimed in Mr. Stetefeldt's paper, *l. c.*, p. 344.

	Gold. Ozs. per ton.	Silver. Ozs. per ton.
Raw sulphurets, ...	3.27	10.80
Final tailings, ...	0.11	8.56

In order to determine the relative importance of the various factors, amount of salt, temperature, and time of roasting, a large number of experiments with muffle roasts were undertaken at the laboratory of the Mining Department of the University, both by myself and by my students under my direction.* These roasts were mainly with the Murchie pyrite already described. Nearly 200 separate experiments have thus been made, from which I select a few of the most reliable and characteristic for illustration.

Table IV shows the combined effect of these three causes. In all these experiments an assay ton of ore was taken, and after the roasting was finished the whole roast was assayed to avoid possible errors due to sampling the roast, loss of weight, etc. An examination of the table will show that within the limit of the salt used (4 per cent.) the loss both of gold and silver increases with the amount of salt used, other things being the same. Second, it is apparent that the effect of time is to increase the loss, but the effect of an increase of temperature on the gold loss is greater than the effect of an increase of time. This is shown by Nos. 2 and 3. Here No. 2 was treated at a dull-red for 5½ hours, but was finished at a light-yellow for half an hour; the loss of gold was 6.46 per cent. Now No. 3 was heated for 9½ hours at a dull-red, and finished for half an hour at a cherry-red, making a total time of 10 hours as against a total of 6 hours in the preceding case, yet the loss of gold is exactly the same. Third, the loss both of gold and silver is greater in all cases where the salt is added after a long oxidizing roast than where the salt is added at the start. This is the general result of all the muffle roasts that have been made. It is invariable in the case of gold, and nearly always the case with silver. The reverse is the case with gold on the large scale where a continuous roasting takes place in the reverberatory furnace, as has been shown by Mr. Aaron. I was at first led to doubt the accuracy of the roasting and assaying, but the fact is undoubted. I shall later give the explanation of this remarkable difference between batch roasts and the continuous methods.

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* Among the students who have aided in this investigation at various times I wish particularly to mention Messrs. Lindley, Russell, Carrol, Hayes, Sutton, and Booth.

TABLE IV.

CHLORIDIZING MUFFLE-ROASTS WITH MURCHIE PYRITE.

Showing Losses Due to Different Amounts of Salt, Temperatures, and Times of Roasting. 1 *Assay Ton* (29.166 *grms.*) *Roasts.*

Conditions of Roasting.					Ounces per Ton.		Percentage loss.	
					Gold	Silver	Gold	Silver
Original Assay-Value of Raw Sulphurets.					4.49	28.39	.....	.....
No.	Salt Per Cent.	Temperature.	Total T.				Per Cent.	Per Cent.
1	2	At start, 7½ hrs. dull red, ½ hr. cherry red...	8	hrs	4.43	27.17	1.34	4.30
2	2	After 2 hrs. dull red, then 3½ hrs. dull red, 1½ hr. light yellow.....	6	"	4.20	24.21	6.46	14.72
3	2	At start, 9½ hrs. dull red, then ½ hr. cherry red.....	10	"	4.20	25.95	6.46	8.59
4	2	After 9½ hrs. dull red, then ½ hr. cherry red	10	"	4.06	25.57	9.80	9.93
5	4	At start, 7 hrs. dull red, then ½ hr. light yellow.....	7½	"	4.17	22.70	7.13	20.04
6	4	After 7 hrs. dull red, then ½ hr. light yellow	7½	"	2.82	17.68	34.97	37.72

In order to study better the effect of time alone on the roasting the experiments recorded in Table V. were projected. The amount of ore taken was as before, 1 assay ton. Six such roasts were weighed out, and the set on the left was mixed with 5 per cent. of salt. This salt had been previously fused, then pulverized, and was kept in a stoppered bottle to avoid losses from the decrepitation of the salt. Six other similar roasts were also weighed out without salt. The two sets were then charged in a pair of muffle-furnaces situated side by side, so that their temperatures could be instantly compared. For the first hour, to avoid decrepitation, covers were used. The room was darkened, and the ore in the muffles was kept as constantly as possible at a dull-red heat, every precaution being taken to keep the temperature and the draft in each furnace exactly the same, a large number of experiments being necessary before all these conditions were reached. At intervals of one hour a roast was taken from each muffle. At the end of 4 hours the two remaining roasts were removed from each muffle, the left-hand set were removed from the dishes when cold, pulverized, and returned to their dishes. The right-hand set received the same treatment, but at the same time 5 per cent. of salt was intimately mixed with the ore. Both pairs were then returned to their respective muffles, and after another hour Nos. 5 from each muffle were removed; and after another hour the last one of each set (No. 6) was removed. The

whole of each roast, together with its roasting-dish, was then assayed by the crucible method. This last precaution was found necessary, because it was found that when salt was used with the ores a certain amount both of gold and silver was absorbed by the roasting-dish. In order then to have them all subjected to the same treatment the roasting-dishes were all assayed with their respective ores. The buttons, with a little adherent matte in the case of the first numbers of each set, were scorified before cupellation.

Table V. shows the original content of the raw sulphurets by scorification assay and the value of the roasts after each succeeding hour of treatment. It also shows the percentage loss at the end of each hour both for gold and for silver, and the percentage loss during each hour.

TABLE V.

CHLORIDIZING MUFFLE-ROASTS TO SHOW EFFECT OF TIME. TEMPERATURE CONSTANT. A DULL-RED HEAT.

Content of Raw Ore: Gold, 4.49 *oz.; Silver,* 28.39 *oz. per ton.*

5 per cent Salt from the Start. 5 per cent. Salt after four hours Oxidizing Roast.

No.	Oz. per ton		Percentages Lost.				Oz. per ton.		Percentages lost.			
			Gold.		Silver.				Gold.		Silver.	
Hrs.	Gold	Silver	Tot'l	Per Hr.	Total	Per Hr.	Gold	Silver	Total	Per Hr.	Total	Per Hr.
			Per Cent.	Per Cent.	Per Cent.	Per Cent.			Per Cent.	Per Cent.	Per Cent.	Per Cent.
1	4.42	27.27	1.11	1.11	3.95	3.95	4.49	27.69	0.00	0.00	2.47	2.47
2	4.39	26.95	2.23	1.12	5.07	1.12	4.48	27.44	0.23	0.23	3.35	0.88
3	4.29	26.65	4.46	2.23	6.13	1.06	4.46	27.13	0.67	0.44	4.44	1.09
4	4.25	26.33	5.35	0.89	7.26	1.13	4.43	26.87	2.45	1.78	5.35	0.91
5	4.17	25.86	7.13	1.78	8.91	1.65	4.08	26 40	9.13	6.68	7.01	1.66
6	4.13	25.37	8.02	0.89	10.64	1.73	3.95	26.25	12.03	2.90	7.54	0.53

TABLE VI.

CHLORIDIZING MUFFLE-ROASTS TO SHOW THE EFFECT OF A CHERRY-RED HEAT AFTER 4 HOURS DULL-RED.

Content of Raw Ore the Same as before in Silver and Gold.

5 per cent. Salt from the Start. 5 per cent. Salt after four hours Oxidizing Roast.

No.	Oz. per ton.		Percentages lost.				Oz. per ton.		Percentages lost.			
			Gold.		Silver.				Gold.		Silver.	
Hrs.	Gold	Silver	Tot'l	Per Hr.	Total	Per Hr.	Gold	Silver	Total	Per Hr.	Total	Per Hr.
			Per Cent.	Per Cent.	Per Cent.	Per Cent.			Per Cent.	Per Cent.	Per Cent.	Per Cent.
5	4.10	24.92	8.69	3.34	12.22	4.96	3.64	23.61	18.93	16.48	16.84	11.49
6	4.08	24.76	9.13	0.44	12.79	0.57	3.55	23.53	20.94	2.01	17.12	0.28

I. *The Gold-Loss.*—(1.) *Five Per Cent. Salt from the Start.*—The loss of gold in this case is evidently pretty constant for the first two hours, being a little over 1 per cent. per hour. During the second hour it has doubled, the cause being evidently the fact, apparent in the roasting, that the chlorine was beginning to be set free in the ore. For the next three hours there is some irregularity in the loss per hour, but it is on an average 1.18 per cent. The average loss per hour for the whole time is 1.33 per cent. The final loss is 8.02 per cent. Evidently here also time is not the most important factor at work; the amount of chlorine set free is the more important cause of irregularity here.

(2.) *Five Per Cent. Salt After Four Hours' Oxidizing Roast.*—The losses of gold for the first four hours' roasting should here have been zero, according to the roasting-tests previously cited. As a matter of fact, in all tests here cited, a small loss of gold actually took place before the salt was added. It was not due, probably, so much to dusting here, as to the fact that the draft of the furnaces had to be so shut off to keep the temperature down, that the chlorine fumes from the salted roasts were drawn into the room, and may have afterwards been drawn into the muffle with the unsalted roasts. In order to avoid dust loss the roasts were not stirred at all.

The effect of the salt addition is, however, manifested by an immediate jump from a percentage loss of gold from 1.78 per cent. to 9.13 per cent., or an increase during the fifth hour of roasting with salt of 6.68 per cent. This enormous increase again shows the effect of a large chlorine generation, for the evolution of the chlorine was here quite rapid, much more so than during any hour of the roasting of the previous set. It again shows how great an effect the rapid evolution of chlorine has on the gold loss. During the last hour the loss of gold, while still greater than during any hour of the previous set, is still less than half what it was during the fifth hour. The explanation again suggests itself that the evolution of chlorine, during this hour, was less rapid than when the salt was first added.

Again, it will be noticed that the final loss of gold with the set where salt was added at the start, is only 8.02 per cent. as against the loss of 12.03 per cent. where the same amount of salt is added at the end of a four hours' oxidizing roast, and is in contact with the ore only two hours in all. The fact is, as already stated, the uniform general result of all the muffle tests that I have made. The

explanation is, undoubtedly, that the loss of gold is due to the amount of chlorine which comes in contact with the gold.

Now, when the salt is added at the start, the chlorine is at first used up as fast as it forms by the sulphur, which escapes as chloride of sulphur. This fact was shown by the covers of the salted roasts, a certain amount of this substance being condensed on them. Of course, as long as the sulphur is present, it protects the gold from attack, and naturally the loss should be less. But when no salt is added till a long oxidizing roast has converted the sulphur into sulphuric acid and acid sulphates, and the salt is then added, the chlorine is then rapidly generated, and, what is more, it all has a chance to act on the gold, which is now no longer protected from its action by the presence of sulphur, etc.

II. *The Silver-Loss.*—(1.) *Five Per Cent. Salt from the Start.*—It is evident that here the greatest loss is at the first hour, being nearly 4 per cent.; after that it is nearly constant, averaging 1.33 per cent., but increasing a little towards the end.

(2.) *Five Per Cent. Salt After Four Hours' Oxidizing Roast.*—Here, too, the maximum loss is during the first hour, being 2.47 per cent.; after that it is also nearly constant, averaging 1.00 per cent.

The great apparent loss, during the first hour of roasting, is probably partly due to an error in assaying, as the crucible results are generally low on ores containing so much sulphur as these which were roasted only one hour, owing to the unavoidable formation of matte, which dissolves to some extent in the slag. It is, however, plain that the silver follows a somewhat different law from what the gold does. Silver is, probably, volatile, both in the shape of oxide and to some extent also as chloride. It is plain that, in both these roasts, after the first hour, the silver loss is nearly proportional to the time of roasting. The fact that the silver loss in the second set (salt after four hours) was less than that with salt at the start, seems to have been accidentally the case in this set; it was more often the other way.

In order to study the effect of increased temperature on the losses, the set of experiments shown in Table VI. was undertaken. This set was roasted under exactly the same conditions as in Table V., except that, for the fifth and six hours, the temperature was raised to a cherry-red heat.

The losses of gold and silver are, in all cases, increased to a remarkable degree.

I. *The Gold-Loss.*—(1.) *Five Per Cent. Salt from the Start.*—During the fifth hour the loss has been 3.34 per cent. as against 1.65 per cent. what it was during the same hour at a low red heat. This doubling of the loss is partly due to increase of volatility owing to the rise of temperature, and partly to the setting free of a greater amount of chlorine; for, during the last hour, we find an increase of the loss amounting only to 0.44 per cent., probably due to the fact that the salt was nearly all decomposed.

(2.) *Five Per Cent. Salt After Four Hours' Oxidizing Roast.*—Here the increase of the gold loss is something enormous, being 16.48 per cent. in the fifth hour as against 6.68 per cent. what it had been for the fifth hour at a dull red heat; the loss is here nearly trebled. The explanation lies evidently not only in the increase of the volatility of the gold due to increase of temperature, but to the sudden generation of larger amounts of chlorine. The sixth hour shows the gold loss to have been only 2.01 per cent. as against 2.90 per cent. what it was for the sixth hour at a dull red heat. The reason lies evidently in the fact that the salt was now more thoroughly decomposed, and that the chlorine supply had begun to fail.

If we compare the final losses for Table VI., we find a loss of 9.13 per cent. of gold where the 5 per cent. of salt is added at the start, as against 20.94 per cent. where the same amount of salt is added after four hours' oxidizing roasting; in other words, the loss is increased two and one-third times.

II. *The Silver-Loss.*—Evidently, in both cases it is enormously increased during the fifth hour, and diminished in the sixth, the evident cause of the latter fact being a failure of the chlorine supply.

The final silver losses are largely increased, particularly where the salt is added after a long oxidizing roast.

The Volatility of Gold in an Atmosphere of Chlorine.

The evident connection between the chlorine supply and the loss of gold naturally suggests that chlorine was the principal cause of the loss. The conflicting statements of various authors on the subject also led to an investigation of the matter.

My first experiments were made in June, 1880. The chlorine used in these experiments was generated from pyrolusite and crude HCl. The gas was passed through a second flask filled with pyrolusite to absorb the HCl, and was then passed in a moist state through a glass tube in which the gold was placed. In my first experiment I used a 30-milligramme fragment of fine gold in a very

porous state, obtained by parting with nitric acid. When chlorine was passed into the tube the gold became discolored, changing to a dark, reddish-brown color; when warmed gradually with an alcohol flame, this discoloration passed away, the gold became bright again, and at the same time a reddish-yellow sublimate, fading off into pale yellow, began to form about an inch from the flame. When the flame was applied to this incrustation, it became first dark brown, then changed into spangles of metallic gold, which, on cooling, were again attacked by the chlorine as before.

The tube was cut in two between the gold and the incrustation, and when the latter was dissolved in a few drops of water, a copious precipitate of metallic gold was obtained with ferrous sulphate.

In another similar experiment, the gold weighed before treatment 23.5 milligrammes, and after one hour and fifteen minutes, weighed only 17.75 milligrammes, or it lost 5.75 milligrammes or 24.47 per cent. of its weight in this time (or per half hour, 8.99 per cent.). In both these cases the gold was in an exceedingly porous state, and exposed considerable surface to the action of the chlorine. There was abundant proof that gold was volatilized in all these experiments. In one case the glass was heated till it began to fuse; the glass was, in this case, colored purple immediately below and around the gold fragment.

Another interesting fact was discovered at this time that I have never seen mentioned before, that the volatilized gold chloride contained in a chlorine stream, when passed into a Bunsen-burner flame, tinges the latter a peculiar and intense green. This I have since found a peculiarly delicate and characteristic test for the presence of chloride of gold in a chlorine stream. That the coloration was due to gold and to no other cause, is proved by the fact that the chlorine stream never, under any circumstances, gave this coloration, except when it had passed over the gold; and that it was not due to copper, apart from the fact that fine gold was used, is evidenced by the fact that, when the gold was replaced by a fragment of pure copper, the volatilized chloride of copper tinged the flame not green, but blue, as is well known to be the case with chloride of copper.

Other work prevented a resumption of these experiments till April of 1888. I then repeated these tests, and obtained others which follow. I placed a clean porcelain dish filled with water over the Bunsen flame through which the chlorine stream, charged with chloride of gold, was passing, and obtained an incrustation of a purple color. This, dissolved in aqua regia, reacted for gold. I also tested the gold flame with a single-prism spectroscope, and ob-

tained a number of lines in the yellow and green, entirely different from those obtained with copper or other metals under similar circumstances. At first I had great hopes of being able to use the spectroscope in furnace work, but at present it does not seem practicable, although I hope to resume this subject at a future time.

I found the most delicate test for the presence of chloride of gold in the chlorine stream, is to make the test in a darkened room, and to turn the stream against the Bunsen flame, so that it strikes against the flame about an inch above its base, and then to view the flame at right angles to the impinging stream against a black background. The colored edge of the flame can then be compared with its uncolored edge, and an exceedingly minute trace of gold can thus be detected, an amount so small, indeed, that, with the spectroscope that I used, no lines whatever could be seen. In this manner I succeeded in detecting the coloration when only 0.17 milligrammes of gold were volatilized by the chlorine stream in half an hour. Unfortunately at a furnace heat, the amount of light set free entirely masks the coloration, and it cannot, therefore, be used in roasting.

I next wished to ascertain whether or not the gold was volatile in an atmosphere of chlorine at all temperatures, in order to account for the conflicting statements that I have already quoted. For this purpose I finally devised the following method of procedure: For all temperatures below the boiling point of quicksilver, the gold was placed in a U-tube of glass, the whole being heated in a steam bath for 100° C., and in an air bath for higher temperatures. The gold used in all cases, was in the shape of buttons of fine gold. This gold was prepared with the greatest care by first parting in the usual manner with nitric acid, then dissolving the gold in aqua regia, diluting to remove traces of silver chloride and evaporating to dryness; the whole was then taken up with water and a little acid, and finally precipitated with oxalic acid, washed out with nitric acid, and finally melted down into two buttons, whose original weights were 1.52990 and 1.84050 grammes, respectively. All the temperatures above mentioned, were determined with an ordinary mercurial thermometer immersed in the bath. I found the actual temperature of the chlorine stream was a few degrees below that of the bath.

For all temperatures above the boiling-point of quicksilver I found it most convenient to place a clear scorifier of white fire-clay into the back of a small gas-furnace, whose temperature could be rapidly and accurately adjusted, or maintained for a long time con-

stant. The gold button was then placed in the centre of the scorifier and the chlorine introduced by means of a long-stemmed tobacco pipe of the English pattern. The bowl of the pipe was placed over the button of gold and the chlorine turned on, and the whole button was thus surrounded by an atmosphere of chlorine at just about the furnace-temperature. In all cases, before the button was added, the pipe and the scorifier were allowed to become thoroughly heated, and the chlorine was passed for some time to volatilize any traces of iron that might be contained in the clay. The clay after treatment always came out as white as snow and completely free from all iron discolorations.* After this was done the chlorine stream was turned off and the gold added to the scorifier. As soon as it had assumed the furnace-heat the chlorine was turned on and the time taken. The temperature was maintained constant, as was also, as far as possible, the strength of the chlorine stream. After half an hour the chlorine was turned off and the scorifier and its contents removed, cooled and the button weighed. The balance used was sensitive to 0.01 mg. The above plan of procedure was found to work perfectly for all the high temperatures, as it could be adapted to secure all the varying conditions necessary in such tests. The button came out clean and free from all adhering impurities, no trouble from adhesion to the bottom of the scorifier being found even when the button had been melted, if a slight shake was given to the latter a moment after it had solidified.

The chlorine in all these experiments was generated from pyrolusite, pure strong HCl being used in this case. The gas produced was passed through a wash-bottle containing a concentrated solution of salt. The chlorine so produced may have contained some traces of ferric chloride, though no evidences of that fact showed themselves. This moist chlorine was used in some of the tests while in others the chlorine was, in addition to the above treatment, passed through a wash-bottle so arranged that the gas passed through about an inch and a half of strong sulphuric acid. This is what is designated "Dry Chlorine" in the experiments, though of course the gas was not entirely free from moisture.

With all the temperatures measured with the quicksilver thermometer a difficulty was found in determining the exact loss of weight

* This fact was so remarkable that a possible application might be made of it in decolorizing fine porcelain at a furnace-heat. The ware seemed, however, to be rendered somewhat more brittle by the chlorine treatment, but this might have been due to the lack of annealing which I had no time for.

which the button suffered, for the gold came out after treatment with a slight brownish film of chloride upon it. This prevented an accurate weight of the button being taken on account of the extreme hygroscopic nature of the film, while if the film were washed off the indicated loss would have been greater than that really volatilized in the given time. As the composition of this chloride was uncertain, and probably not constant, no exact allowance could be made for it. The fairest method of approximation appeared to be to weigh the button as rapidly as possible, then wash and ignite it, and when cold weigh again then to take the mean of these results. The following, which was the extreme case of this difficulty, will illustrate:

"*Experiment No.* 11." *Time* ½ *hour. Temperature* 250° *C. Chlorine stream moderate.*

Original weight of gold before experiment		1.42466 grms.			
Weight after with chloride film, etc.	(a)	1.41117 "	Loss of w't	0.01349 grms.	
" " washing and ignition,	(b)	1.41016 "	" " " "	0.01450 "	
Difference between (a) and (b)		0.00101 "	Mean loss taken	0.01399 "	

The latter figure, 0.01399, is the one given in the tabular resumé that follows. The error introduced by this cause is evidently a very slight one. With all the high temperature experiments there was no difficulty from this source, for the gold came out free from any film whatever, and all that was necessary was to weigh the button, and hence in these results there is no uncertainty from this cause.

In all the low-temperature experiments where glass tubes could be used it was easy to get most indubitable proofs of the volatilization of the gold other than those obtained by loss of weight; the flame test already referred to was obtained in all cases except at 100° C., where the loss was only 0.01 mg. in half an hour; here no indication could be obtained with the flame. In all cases there was a sublimate of chloride of gold in the cooler portions of the tube, showing plainly that the chlorine deposits a part of the chloride of gold when a stream of chlorine saturated with it at a certain temperature is cooled. With damp chlorine the sublimate was in the form of a rich ruby-colored liquor fading to a pale yellow, but with dry chlorine the tube was filled with a net work of beautiful ruby-colored crystals which rapidly deliquesced in the air.*

* Some of these crystals were examined in the microscope. Great difficulty was had in mounting them, without decomposition from heat on the one hand or the reducing action of the mounting substance on the other. The method which was finally selected was to place them on a glass slide on which a previously prepared dried ring of asphaltum had been placed; a cover was then placed over the ring and

In some of the high-temperature experiments additional proof of the volatilization of the gold was obtained by using an aspirator tube of hard glass, one end of which was introduced into the muffle near the pipe bowl; the other end was then connected with a filter-pump and the fumes thus drawn through the tube a deposit was found at the hot end of the tube several inches long of a reddish-brown color. This, dissolved in aqua regia, reacted for gold. Also in the case of the experiments conducted at a melting heat, a deposit of a similar color formed on the front of the furnace above the mouth of the muffle. This deposit also contained gold, as also iron from the clay of the muffle itself. It recalls to mind the deposit mentioned by Mr. Aaron in the passage already quoted, where he was thus led to discover the true cause of the losses he was suffering in roasting. Besides, there was always present when any amount of gold was volatilized, a deposit of a purplish color immediately outside of the bowl of the tobacco pipe, and also on the scorifier immediately outside of the bowl. In one case, that of "Expt. 12 a," the scorifier was placed just outside the mouth of the muffle, where a thermometer placed beside the pipe-bowl indicated a temperature of from 180° to 250° C. or an average of 205° C. When this scorifier was withdrawn after a half hour's treatment it presented a remarkable appearance. On the hot side, toward the interior of the muffle, was a beautiful fan-shaped deposit of minute crystals of metallic gold. These, afterwards examined under the microscope, proved to be microscopic octahedrons and apparently dodecahedrons. On either side of the gold button, and extending in a direction parallel to the front of the muffle was a narrower fan-shaped stain or incrustation of a purple color, these again shaded off into broader fan-shaped stains of a dark-blue color, while between these two and on the coldest side was a pale-yellowish stain. Evidently, as the chloride of gold escaped into an atmosphere containing less chlorine, it had been decomposed by the different amounts of heat differently.

With this explanation, the results recorded in Table VII. will be intelligible.

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the whole gently heated till the cover adhered, thus making an air-tight compartment. The crystals were partly melted by the heat in mounting but reformed in minute microlytes of a straw-yellow color, while the main mass of crystals retained its ruby-red color. In some of the pale yellow crystals minute twin crystals and crystalline aggregates of metallic gold were visible with powers magnifying 500 diameters. The crystals of gold chloride were monoclinic.

TABLE VII.

Tabular Resumé of Experiments on the Volatility of Gold in Chlorine Gas.

Temperature.	No.	Orig. Wt. of Gold, Grms.	Actual Wt. Lost ½ hr.		Calc. per cent loss 1 Gm. Button.	Remarks.	Exp. No.
			Grms.	Per Cent.			
					Per Cent.		
98° to 100° C.	1	1.40564	0.00001	0.0007	0.0008	Strong cur. dry Cl. steam bath.	14
100° to 130° C.	2	1.43030	0.00017	0.01	0.013	Flame test for gold, air bath.	8
150° C.	3	1.43013	0 00056	0.04	0.044	Flame test for gold, " "	9
200° C.	4	1.42957	0.00408	0.28	0.32		10
205°? [180° to 230°]	5	1.84050	0.00800	0.43	0.53	Done in muffle, Temp. uncertain.	12a
210° to 220° C.	6	1.40563	0.01324	0.94	1.08	Good stream Cl. air bath.	15
250° C.	7	1.42466	0.01399	0.98	1.11	" "	11
300° C.	8	1.41016	0.00278	0.197	0.22	" "	12
Over 300° C.	9	1.43136	0.00106	0.074	0.08	Done in muffle, Temp. uncertain.	7
Below Red Heat	10	1.43161	0.00025	0.017	0.02	No subl. vis. in aspirator tube.	6
Incip. Red Heat	11	1.68930	0.00073	0.043	0.052	Dry chlorine. In muffle.	23
" " "	12	1.63316	0.00086	0.053	0.062	" " " "	30
" " "	13	0.89346	0.00091	0.102	0.098	Treated in same muffle with 12.	31
Low Red Heat	14	1.40726	0.00162	0.12	0.13		13
Cherry Red Heat	15	1.47242	0.00357	0.24	0.28	Moderate current chlorine.	3
" " "	16	1.46885	0.00343	0.23	0.27	" " "	4
" " "	17	1 63693	0.00377	0.23	0.27	Treated side by side, in same atmosphere, dry Cl. same temp.	28
" " "	18	0.89652	0.00215	0.24	0.23		29
" " "	19	1.68857	0.00184	0.11	0.13	Very weak cur. Cl.	24
" " "	20	1.68673	0.00323	0.19	0.23	Stronger " "	25
" " "	21	1.63230	0.00404	0.25	0.29	Side by side with following.	32
" " "	22	0.89346	0.00376	0·43	0.39	Draft in muffle stronger here.	33
Incip. Yellow Heat	23	1.69607	0.00677	0.40	0.48		22
Just Below Straw Yellow Heat	24	1.68350	0.00811	0.48	0.57	Very weak current Cl.	26
" " " " "	25	1.67439	0.03422	2.04	2.43	4 or 5 times as much Cl as preceding experiment.	27
Just About Melting	26	1.71683	0.02076	1.21	1.45	Moderate Current Cl.	21
Above Melting Heat	27	1.46541	0.03380	2.31	2.62	Gold cond. in aspirator tube.	5
" " "	28	1.83264	0.04284	2.34	2.86	" " " " dry Cl.	16
" " "	29	1.78980	0.03656	2.03	2.47	Impinging stream dry Cl.	17
" " "	30	1.20605	0.03305	2.74	2.92	Atmosphere dry Cl.	18
" " "	31	1.75344	0.03661	2.07	2 51	" " "	19
" " "	32	1.52990	0.05730	3.74	4.32	" damp Cl.	1
" " "	33	1.47260	0.00018	0.01	0.014	No chlorine present. Temp. as in 32.	2
Near a White Heat	34	1.17300	0.06069	5.17	5.46	Strong current dry Cl.	20

The first column contains the temperature, the second the reference number which will be used in the discussion of the results which follows. The actual order in which the experiments were performed is given in the last column, headed "Expt. No.," which I have retained for convenience in referring to my detailed notes, but I have here preferred to group the results according to the temperatures, to bring out more clearly the relation existing between the temperature and the volatilization loss. The third column gives the original weight of the button in grammes before each experiment, and the fourth and fifth give the actual loss of weight in grammes and the percentage loss suffered by the button in half an hour. The sixth column has been calculated for the purpose of bringing the buttons to a standard of comparison. For this purpose, I have selected an ideal gold-button weighing one gramme. I have assumed, as I shall shortly show to be practically the case, that the loss is, other things equal, proportional to the surface exposed to evaporation. Now, the surfaces of solids of the same composition and shape is proportional to the two-thirds power of their weights, and as the surface of one of these buttons is nearly constant during the short time of one of these tests, the reduction to standard, or to the weight that would be lost in the same time, and under the same conditions, by a button weighing one gramme, is easily made with sufficient exactness by dividing the weight actually lost by the button by the two-thirds power of its initial weight in each case. The percentage loss, reduced to standard as given in column six, easily follows by simply pointing off. The standard percentage loss is, of course, always greater than that of the actual percentage loss of the experimental button, when the latter is larger than one gramme, and *vice versa*.

The column headed "Remarks" gives the amount of chlorine used. When no remark is made, a moderate current of four to six litres per hour is meant; "weak" means less than that; "strong" means more. The generator was frequently gauged by measuring the stream for a minute, and afterwards judging of the current by the number and size of the bubbles in the wash-flask.

A careful study of the facts recorded in the table will, I think, establish the following conclusions:

(1.) *Effect of Temperature.*—At 100° C., the volatility of gold in an atmosphere of chlorine is almost zero (No. 1). That the loss begins, above this temperature, to rapidly increase to a maximum at a temperature of about 250° C. (for a one-gramme button, No. 7

shows the loss to become over 1 per cent. in half an hour.). That it rapidly diminishes to a minimum at a temperature somewhere below a red heat (No. 10 shows a standard loss of 0.02 per cent.). That it again increases, but more slowly, to another maximum at a temperature above a melting heat, and that this increase is apparently continuous between a red heat and a white heat.

The ratio of losses at various temperatures is also a very instructive study. At incipient redness, the standard loss is already 0.05 per cent. (or 0.052 milligrammes on a one-gramme button) in half an hour; at a low red it is double that; at a cherry red it is five to seven times as great as at incipient redness; at incipient yellow it is more than eight times what it is at incipient redness; while at a melting heat it is nearly thirty times as great.

These facts explain the extremely great importance of a proper regulation of the temperature in the chloridizing roasting of all silver-ores that contain gold, and, in fact show, as I have already insisted, that a low temperature is even more important than a short time, as far as the gold loss is concerned. The rather anomalous relation of gold in an atmosphere of chlorine at various temperatures, also offers a probable explanation for some of the contradictory statements that have found their way into the literature of the subject.

In order to determine the effect of heat alone in causing a volatilization of gold in the absence of chlorine, Nos. 32 and 33 were among the first experiments undertaken. They were both conducted at the same temperature, above a melting heat, and for the same time. The only difference in the treatment in the two cases was that No. 32 was conducted in an atmosphere of damp chlorine, while No. 33 was conducted in the open muffle in an atmosphere of air, no chlorine being present. The actual percentage loss was 3.74 per cent. with the chlorine, and 0.01 per cent. without. Or, reduced to standard, the losses were 4.32 per cent. with chlorine, and 0.014 per cent. without. In other words, the chlorine at this temperature caused the volatilization loss to become 308 times what it was in the air. At temperatures below melting the losses of gold in the air were too inconsiderable to show themselves in half an hour.

(2.) *The Effect of Strength of Chlorine Stream.*—Nos. 19 and 20 were undertaken to determine this point. No. 19 was carried on with a very weak current of chlorine. No 20, with a stronger one, was carried on for the same time at the same temperature, a cherry-red heat; the loss in the first case was about half that in the second.

No. 24 was carried on at a temperature just below a straw-yellow with a very weak current of chlorine, and No. 25 for the same time and at the same temperature with a current perhaps four or five times as great. The loss in the latter case was nearly five times as great as in the former. With these exceptions, and those of Nos. 1, 33, and 34, the chlorine currents were kept as nearly as possible of average intensity.

Evidently, the amount of gold-loss is increased by an increase of the chlorine stream, other conditions remaining the same. Probably the loss will be found to vary directly as the volume of chlorine passed, provided pressure and temperature are constant, and sufficient time is allowed for the chlorine to become saturated at the given temperature.

Another proof of the loss being dependent on the strength of the chlorine stream is by Nos. 17 and 18. Here the two buttons were placed side by side in the same furnace and in the same atmosphere; the losses, reduced to standard, proved less for the smaller than for the larger button. The reason for this fact was that there was a crack on the side of the muffle near the larger button, and consequently the draft was stronger there and more chlorine was drawn out on that side. In Nos. 21 and 22 the positions of the buttons were reversed with reference to the crack, and the relative losses reduced to standard are also reversed.

(3.) *Effect of Surface.*—Theoretically, the losses, other conditions remaining the same, should be proportional to the surfaces exposed to evaporation. Experiments Nos. 17 and 18 and also 21 and 22 were originally undertaken to determine whether or not this were true. If this were the case, the percentage losses, reduced to standard, should have been the same when temperature and strength of current were the same. We have seen that when thus reduced the losses were not the same, but the strength of current was different, being stronger on one side than the other; but the position of the large and small buttons being reversed in the two pairs of experiments, the effect of this cause should be eliminated from their average. The losses of the small button, in the two experiments, reduced to standard and averaged, was 0.31 per cent., and those of the large one was 0.28 per cent. When we reflect how difficult it is to avoid slight differences in temperature, draft, etc., and also that owing to slight differences in size the shape of the buttons is not quite the same, this agreement is sufficiently close to justify us in assum-

ing that the volatilization loss is, other things equal, proportional to the surface exposed.

Now when we remember the extreme minuteness of the gold in many pyritic ores, it is easy to understand the extreme rapidity with which the gold may be attacked in roasting these ores with salt. Thus we have seen that (No. 14) a one-gramme button loses 0.0013 grammes in weight at a low red heat in half an hour. Suppose now that this same gramme of fine gold were subdivided into small spheres, each weighing 0.01 mg. The weight of the one-gramme sphere would be to that of the small ones as 100,000 to 1, while its surface would be as that of a small one, as 100 to 1. But, as it would require 100,000 of the small ones to make the weight of one large one, the total surface exposed to evaporation would be 1000 times as great with the small spheres as with the large one. Hence, while the single-gramme sphere was losing 0.0013 grammes, the gramme in the finely-divided state would lose (if the rate of loss remained unchanged) 1.3 grammes, or, in other words, if exposed in an atmosphere of pure chlorine, under the circumstances mentioned, at a red heat, the finely divided gold would have been entirely evaporated before the half-hour was up.

In actual practice, many circumstances would reduce this extreme rate of loss. The surface exposed would be constantly diminishing, and hence the rate of evaporation would not be constant. The chlorine would probably finally become saturated with gold-chloride, and would take up no more, etc., and in roasting, an atmosphere of pure chlorine is probably seldom produced. Still, it is easy to see how important an influence a finely-divided condition of the gold has on the gold-loss. It is easy to understand how Mr. Stetefeldt could obtain a volatilization of 93 per cent. of the gold-content of the ore he treated, simply on account of the fine state of subdivision in which it existed and its low grade.

The above assumption as to the size of the gold particles in auriferous pyrites may seem, at first sight, unreasonable; but when we remember that the battery-screens usually retain the coarse gold, and the plates catch most of the rest, it is not so unreasonable as at first sight may appear. As a matter of fact, I have carefully washed, on a batea, samples of concentrated sulphurets from many California gold mines, and have often been unable to find any signs of free gold, even when the concentrates were examined with the microscope. When these same sulphurets were roasted, however, the same method of treatment would usually show fine particles of metallic gold when

the concentrates were thus examined. The roasted samples of Murchie ores (roasted without salt) thus yielded numerous small particles of free gold. The smallest of these, which was nearly round, measured 0.025 millimetres; assuming it to be spherical, its weight would be only 0.00016 mg. The largest that I found was 0.156 mm. long by 0.052 mm. wide, and its thickness was probably much the same. Assuming it to be equivalent to a sphere with a diameter of 0.1 mm., we should have as the weight of this particle only 0.00995 milligrammes, or an amount that a very sensitive assay-balance would barely indicate.*

Of course, the above remarks are not intended to apply to all auriferous sulphurets, as the contrary is often the case as to the size and distribution of the gold; but it will, perhaps, serve to explain why some ores suffer an extraordinarily greater loss than others of the same richness and apparent composition, when roasted in the same manner and with the same amount of salt.

The Bearing of the Miller Process on the Above Results.

I come now to a consideration of the method of refining alloys of gold without appreciable loss, as first practiced on a small scale by Thompson and Aiken, in 1838, and since successfully introduced on a working scale at the Australian mint by Mr. Miller, and generally known as the Miller process. The remarkable success of this process on a working scale, the total losses from all causes amounting only to 0.011 to 0.019 per cent. of the weight of the gold treated, would seem to invalidate the conclusions that I have drawn from my own experiments. But, when the process is carefully studied, it serves rather to confirm than to contradict them.

First, regarding Mr. Thompson's work: No statement of the weight of the gold experimented upon, its surface, or the time of exposure to the chlorine is given in the abstract published by Prof. Percy; and it is easily possible that a loss due to the volatilization of the fine

* I came across the curious fact in some of the above experiments, that when two of the gold-buttons used came in contact, ever so lightly, at a temperature considerably below a red heat, they were welded together at the points of contact so firmly that it took considerable force to separate them. This recalls to mind the similar welding that takes place so often in the annealing-cup when parted fragments of gold are warmed to a red heat. It seems not unlikely that the particles of gold that I measured with the microscope in the roasted ore, existed in the raw sulphurets in an even more finely-divided state, and were thus welded together and agglomerated in the act of roasting. The shape of some of them would seem to indicate that fact.

gold may have taken place, in a short time of exposure, and with small amounts of gold which his balance may not have been sensitive enough to indicate. Besides, Mr. Thompson had evidently made up his mind the loss did not take place, and no doubt did not push matters very far to establish a point he already regarded as settled. Further, Mr. Aiken states that the method of procedure used was to "put the gold alloy into a small porcelain tray, with a little chalk or common salt." This was then carefully introduced into the hottest part of a porcelain tube, stated to be white-hot in some cases, and the chlorine passed through. He goes on further to say that:

"On examining the contents of the tray, after the production of the vapor had ceased, the button of gold was found imbedded in a melted mass of sodium chloride (or chloride of calcium if chalk had been put in the tray) mixed with chloride of silver, *the presence of the alkaline chloride seeming to have the property of preventing the volatilization of chloride of silver.*"

This last sentence is a good illustration of how near one may come to a discovery and miss it. If Mr. Aiken had said instead, that the presence of the alkaline chloride, together with that of the chloride of silver, seems to have the property of preventing the formation and volatilization of the chloride of gold, he would have given a true explanation of the facts. We shall see this more clearly when we come to examine a little into the Miller Process.

This process is very clearly and critically described in Prof. Percy's *Metallurgy of Silver and Gold*, Part I., pp. 405–437. In brief, the chlorine is passed to the bottom of a crucible filled with molten gold by means of a clay pipe-stem. The gold to be refined contains several per cent. (an average of about 10 per cent.) of silver together with small quantities of base metal. The gold is usually treated in lots of 600 to 700 ozs. at a time. The molten gold is always covered by a layer of molten borax glass, which Mr. Miller regards necessary to prevent the volatilization of the *chloride of silver*. The chlorine is rapidly absorbed by the base metals and the silver, which latter floats above the molten alloy and under the borax cover.

The process is further described in detail thus (Percy, p. 415): "When the chlorine is first passed into the alloy dense fumes of the base metals escape from the lid of the crucible, which consist of the volatile chlorides of some of the baser metals, and not of chloride of silver, and which are particularly dense when much lead is present in the gold. When a cold body is placed in these lead fumes a

white deposit is formed upon it. After a time, which varies with the proportion of base metals in the gold, the fumes cease. So long as any sensible quantity of silver remains in the molten gold the whole of the chlorine continues to be absorbed, and the quicker the supply the shorter the operation. When the operation is nearly over fumes of a darker color than those produced at the beginning make their appearance, and the end of the refining is indicated by a peculiar flame, or a luminous vapor, of a brownish-yellow color, due to the escape of free chlorine at this period [probably colored by NaCl, Fe_2Cl_6, etc.], as may be seen by removing the plug from one of the holes in the crucible-lid. But this appearance is not of itself a sufficient indication that the operation is finished, *for it is not completed until this flame imparts a peculiar reddish or brownish-yellow stain to a piece of tobacco-pipe or similar substance when it is held in it for a moment, and so long as any other color is thus imparted to the piece of pipe the refining is unfinished.*"

The italics are mine. No one seems to have determined the composition of this peculiar stain. From the description I should strongly suspect that it contained gold, as its appearance agrees very well with those I obtained in my own experiments. It is altogether probable that had the treatment been pushed beyond this point the loss of gold would have increased very rapidly.

There is another fact in connection with this interesting process that bears upon this question. It is found that the average of the Australian gold, when refined in this manner, always leaves as much as 2 per cent. of the gold with the chloride of silver which floats above the molten gold under the borax cover. When this chloride of silver is afterwards reduced by metallic zinc to the metallic state, and then dissolved in nitric acid, the gold remains behind in a finely divided flaky state, and not as rounded globules, as Mr. Miller reasonably suggests would have probably been the case if the gold had been mechanically thrown up into the fused chloride of silver by the passage of the chlorine current. Prof. Thomas Price, who tried some of the California gold bullion by this method on a working scale at his laboratory in San Francisco, informs me that with the California gold, which generally contains more silver, he found this amount of gold carried by the chloride of silver to amount to 5 per cent., and even 10 per cent. of the total weight of the gold.* These facts tend to indicate that the

* For this reason and on account of the large amount of silver bullion in the San Francisco market requiring parting, he states that the Miller process, while technically successful with California gold, is hardly able to compete commercially with

gold exists in the fused mass of chloride of silver as a double chloride of silver and gold. The latter not being volatile, partly on account of its combination with the less volatile chloride of silver, and partly because both are covered with a layer of fused borax-glass, through which its vapor has not sufficient tension to penetrate in any great quantity.

It is evident that the conditions for volatilizing the gold are less favorable in the Miller process than in the experiments that I conducted. For, in the first place, the presence of the silver and the base metals must act to protect the gold as long as they are present, for the affinity of chlorine for these metals is greater than that for gold. Moreover, the gold produced by this process is never absolutely fine. Percy states that the refined gold averaged 993.5 fine, the remaining 6.5 thousandths being silver. It would be interesting to know what the effect of pushing the refining to absolute fineness would have on the gold-loss. An appreciable loss would then probably show itself. Next, the surface of the gold exposed in the Miller process, is incomparably less in proportion to the mass treated than in my experiments. For in a crucible holding 600 to 700 ozs. (19,000 to 22,000 grammes) of gold the chlorine can only come in contact with the gold by bubbling through it, and it does this only after the base metals and the silver have been removed.

In order, however, to make a more definite comparison, let us see what the loss would be if it were possible to maintain a weight of say 20,000 grammes of fine gold in the form of a sphere in a molten state in an atmosphere of chlorine, so that the conditions would be similar to those of my experiments. Now the gold sphere of 20,000 grammes would have a surface 736.8 times as great as a 1 gramme sphere, or that of a button of 1 gramme of similar shape. The average of all my experiments above a melting heat, except the last (which was at nearly a white heat), gives for a button of 1 gramme a loss of 3 per cent., or 0.03 grammes in half an hour, hence the evaporation from a button of 20,000 grammes would be 736.8 times as great, or 22 grammes, and as the Miller process lasts about an hour and a half, with gold containing 10 per cent. of silver the loss would have been on fine gold in the same time 66 grammes, or 0.33 per cent. of the weight of the gold operated upon. Now, when we

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the ordinary silver quartation process. He suggests that it might be well adapted for refining the nearly pure gold produced at the chlorination works where chlorine is at hand and the other methods of refining are not convenient.

consider that the silver* and the base metals, so long as they are present, must protect the gold; that before the silver is all gone the process is stopped; that the surface exposed to the action of the chlorine is less in the Miller process; that no chlorine escapes from the pot except towards the end of the process, and that all through the operation a layer of molten chloride of silver, and above that a layer of molten borax, covers the surface of the gold, it is not difficult to understand why the total loss of the process is reported to be only 0.011 per cent. to 0.019 per cent., including sweepings, etc. On the other hand, the chlorine acts under considerable pressure in the Miller process, as it has to force its way through the molten gold. This probably accounts for the large amount of double chloride of silver and gold.

Finally, that some of the gold as well as silver is actually volatilized in the Miller process in spite of all the consideration mentioned is evidenced by the fact that the dust from the short horizontal flue was found to assay:

Gold, ...	0.441 oz.
Silver, ...	0.853 oz.

I have considered thus at length the principal facts of the Miller process, because they at first sight appear to contradict the results that I have obtained. I think I have shown that they do not. And while the numerical values that I have found will probably be somewhat modified by future and more extensive experiments I am convinced that the general conclusions that I have drawn will stand.

The Effect of Chlorine on Other Metals at a Cherry-Red Heat.

In order to study the effect of chlorine on some of the other metals that accompany gold, the following experiments were conducted at a cherry-red heat in the muffle on the same plan as those conducted in the muffle with gold.

Silver.—A button of silver weighing 0.60118 grammes was exposed in a scorifier under the bowl of a new long-stemmed pipe to an atmosphere of chlorine for half an hour at a cherry-red heat, and, therefore, below the melting-point of silver. The silver button became immediately coated with a film of chloride of silver which was

* 2000 grammes of silver being present.

fused and of a dark-brown color. It rapidly increased in thickness, and at the end of half an hour the silver button had greatly decreased in size, but was still contained in the centre of the button of fused chloride of silver. On removing the scorifier from the furnace the chloride remained fused for some time, and it was easy to obtain the button of silver nearly free of chloride by letting it roll out of the chloride which remained behind. When the adhering chloride was dissolved off the silver button weighed 0.270 grammes, or, in other words, 53.4 per cent. of the silver had been changed into chloride in this time. Evidently, if the button had been smaller, the conversion would have been complete even in this short time. The chloride of silver when cold was colorless and transparent, as usual. Very possibly the chloride of silver when molten has the property of dissolving chlorine; at any rate the chlorine must have been transmitted through a layer of chloride of silver an eighth of an inch thick. Very possibly this molten state of the chloride is the condition necessary for the thorough chlorination of silver in roasting silver ores, and the absence of this condition the reason why silver cannot in many cases be so readily chloridized in the cold. There were evident signs of volatilization of the silver, for the scorifier was coated with a film of a delicate rose or pink color. The amount of this loss was not determined.

Copper.—A button of fine copper, weighing 0.39570 grammes, was treated exactly as above, at the same temperature, etc., in a moderate stream of chlorine. Copious fumes at once issued from the scorifier; these were of a brownish-red color as they rose against the cherry-red background of the muffle walls, but they were of a bluish-white color as they issued from the mouth of the muffle into the open air. They deposited on a cold surface of fire-clay a copious film of a pale greenish-yellow when thin, and of a deep, rich orange-brown when thick. The whole film instantly disappeared when a drop of water was allowed to fall upon it.

In fifteen minutes the copper button had entirely disappeared, and in its place was only a small, dark-colored drop of molten copper chloride. This, when removed from the furnace, partly evaporated in bluish-white fumes, and partly remained in the scorifier as a black crystalline film which, when cold and wetted with water, turned green.

Iron.—I next took an ordinary wrought-iron tack, which weighed 0.50674 grammes, and treated it in the same manner as the copper at a cherry-red heat, in a moderate stream of chlorine. The iron

was also instantly attacked, a dense reddish-brown vapor issuing from beneath the pipe-bowl; this, as it issued from the muffle into the air, still retained its color, and where it deposited itself on cold fireclay, it was still of the same rich, reddish-brown color. Immediately around the outside of the pipe-bowl and on the inside of the scorifier was a bluish-black deposit of minute crystals of magnetite gradually shading off into a reddish deposit of iron-oxide, in which no crystalline form could be discovered with the microscope. 70.3 per cent. of the iron was volatilized in half an hour.

Lead.—I next tried lead in the same way, and found a dense cloud of whitish vapor of chloride of lead issue from the furnace, which was still white when deposited on cold surfaces.

The action of chlorine on antimony, arsenic, sulphur, etc., being already too well known to require further experiment, none were made on these substances.

Mr. Stetefeldt, in referring to the losses he found in roasting, in the paper already cited, seems inclined to throw the responsibility of the volatilization of the gold on the presence of chloride of copper, although he states also that the presence of the copper does not appear to be essential. Now it is, of course, possible that the volatilization of the gold is aided by the presence of the more volatile chlorides of copper, iron, lead, etc.; but, in either case, it is the chlorine which is the real cause behind the formation of all these chlorides, and it seems to me that the presence of the copper, iron, lead, antimony, arsenic and sulphur, on account of the greater volatility of their chlorides, and of their greater affinity for chlorine, should rather serve to protect the gold from the attack of the chlorine, and thus serve to reduce its volatility.

At the same time, when no other source of chlorine is present, and these salts are in contact with the gold, it is not improbable that they would share some of their chlorine with the gold, and thus cause its volatilization also. Further experiments are, however, necessary to clear up this point.

The Condensation of Chloride of Gold from a Chlorine Stream.

I have already quoted the statement of Mr. Aaron, that the bulk of the material caught in his dust-chambers was not richer than the ore treated. On inquiry, I find this to be generally the case; those who have erected dust-chambers, with a hope of effecting a great

saving in this respect, have been usually disappointed, as the dust often assays even less than the ore, and is seldom much richer.

Many metallurgists regard this as a proof that no gold is lost in roasting. This conclusion seems hardly warranted, in the light of the experience recorded by Mr. Aaron, and of the fact that I have frequently found the flue-dust from furnaces, where I know a very large gold-loss to have occurred in roasting, to assay very much as the ore did. The true conclusion is, as was, I believe, first suggested by Mr. Aaron, *that the gold is volatilized in a form not easily condensible.*

In my experiments with chloride of gold in the chlorine stream, I have already pointed out that a considerable amount of chloride of gold is condensed when the temperature of the stream is reduced, but I have never been able to condense the whole of it by cooling alone. In fact, I have found it impossible to remove the chloride of gold entirely, so long as the chlorine was present. It seems that, as already quoted from Pratt, a higher and more volatile chloride of gold than $AuCl_3$ forms under these circumstances.

Thus, no matter whether I allowed the stream to pass through a long, glass tube exposed to the cooling action of the air, bubbled it through wash-water, passed it through charcoal dry and wet, through dry iron nails, or through nails dampened with water, I was able to get more or less of the flame-test for gold. The two latter methods with wet charcoal and wet iron nails, however, were effective in removing some of the gold.

But I found that, in order to remove the gold with any degree of efficiency, it is necessary to destroy the chlorine which carries the chloride of gold. Anything which effects this result, will do. I have tried several, with more or less success. Sulphydric acid does very well, but there is some danger of explosive union of this gas with chlorine, and a considerable quantity of very disagreeable chloride of sulphur is produced, which would render the subsequent extraction of the gold very unpleasant. The reaction, which takes place between chlorine, sulphurous acid and water, whereby sulphuric and hydrochloric acids result, may be easily taken advantage of, if sufficient time and surface are provided for the reaction to take place efficiently.

In my experiments I found it convenient to use, as a condensation-tower on a small scale, a glass jar, 9 inches high by $2\frac{1}{2}$ inches in diameter. Into the bottom of this I introduced a glass tube bearing the chlorine stream charged with gold; by means of another tube

I introduced a stream of sulphurous acid generated from copper and sulphuric acid. A tubulature near the bottom allowed water to be added for the gases to bubble through, or for the withdrawal of the same from time to time.

During two hours I passed a very small amount of gold into the condenser, in all:

Sent to condenser, ...		4.83 mg. of gold
Metallic gold filtered from condenser water,	3.46 mg.	
Gold precipitated from condenser water by H_2S,	0.15	
Total gold condensed, . .		3.61 mg. or 74.74 per cent.

In this case, however, whenever the stream of chlorine was much increased, the gold flame began to show itself, showing that sufficient time and surface were not provided for the reaction of the gases to take place.

I next used charcoal, with which the cylinder was filled. This was wetted with water, which was allowed to trickle constantly through it, being withdrawn at the bottom, and replaced from time to time. The gold-bearing chlorine stream and sulphurous acid were introduced, as before, at the bottom of the "tower." In all cases considerable heat was generated by the reaction, the contents of the condenser being rapidly heated nearly to 100° C. In one case the condenser was only partly filled with charcoal, and during the experiment nearly all the gases that were passed to it, were absorbed by the water and the charcoal, only a very slight amount of gas of any kind escaping, and no sign of chloride of gold appearing by the flame-test, except when the sulphurous-acid stream was reduced at the same time that the chlorine stream was increased. So long as the sulphurous acid was in excess, no sign of gold-flame appeared. After the experiment was over, the charcoal was drained on a filter and burned on roasting-dishes. Copious fumes of sulphuric acid and hydrochloric acid were set free at first, and considerable decrepitation of the charcoal took place, leading, no doubt, to a mechanical loss of gold. During the condensation which lasted several hours, there had been:

Volatilized, including that which condensed in delivery tubes, which was accidentally lost and, therefore, could not be deducted to give actual amount to be charged to the condenser, . .	21.92 mg. of gold.
Obtained from charcoal ash, ...	14.24 " " "
Losses (condensed in tube, not condensed, etc.),	7.68 " " "

Or, there was actually saved of the total amount volatilized, 64.92 per cent. If the amount condensed in the tube before it reached the condenser, had been deducted, the percentage yield of the condenser would have been still higher.

In another experiment, the condenser was filled with charcoal, 100 grammes being required for that purpose, and the apparatus was fed with a gold-charged chlorine stream for eleven hours, sulphurous acid and water being added as before. No signs of a gold-flame were detected during the experiment. The charcoal in the lower part of the condenser became coated with metallic gold.

In roasting this large amount of charcoal, in spite of the greatest care in roasting, considerable decrepitation took place, and no doubt some of the gold contained on the outside of the charcoal, was thus lost.

Sent to condenser in eleven hours, . . .	67.72 mg. of gold.
Obtained from the ash of charcoal, . . .	45.58 " " "
Loss from all causes, . .	22.14 " " "
Actual yield of condenser, . .	67.31 per cent.

Coke was tried in the condenser instead of charcoal, and appeared to work equally well in the condenser, and had the advantage of roasting quietly without any decrepitation. Time did not suffice to push these experiments with coke to a certain conclusion.

Iron was also used in the condenser. In this case the action on the gold is also very effective, provided sulphurous acid and water are present. Considerable heat is evolved, ferrous sulphate and chloride are formed, and usually a little sulphur is set free, probably from the reduction of some of the SO_2 to H_2S, and its subsequent reduction by reaction with SO_2. Very little gas escaped except a little hydrogen which might, indeed, give rise to explosions on the large scale, unless it were provided against.

In one case cast-iron turnings were used. After passing the chlorine stream charged with gold for several hours, no sign of gold-flame being perceived during that time, the iron turnings were dissolved in dilute sulphuric acid which took several days, the residue containing graphitic carbon roasted, and then the whole residue assayed.

The total gold volatilized was, . .	46.32 mg.	
Gold condensed on walls of delivery-tube,	28.60 "	
Sent to condenser, ...	17.72 "	
Actual yield of iron from condenser, .	13.94 "	78.67 per cent.
Losses, . . .	3.78 "	

In another experiment I took 200 grammes of wrought-iron nails passed in SO_2 and water, and caused the ferrous sulphate and chloride to circulate, by withdrawing it from the bottom of the apparatus and allowing it to drip down from the top from time to time.

The total amount of gold volatilized was	46.98	mg.
Chloride of gold condensed in delivery-tube contained gold,	26.00	"
Passed to condenser, ...	20.98	"
Weight of gold obtained after dissolving iron nails, .	17.98	"
Losses due to all causes, .	3.00	"

Or, in other words, the actual saving effected by the condenser was 85.70 per cent. of the gold sent to it. It is evident from these results that a greater percentage of the gold caught might have been effected by either of these means, if a greater amount of contact surface had been supplied.

The following remarks apply to either of the above described methods. Sufficient surface must be supplied for the contact of the gases, the sulphurous acid must be in excess of the chlorine, and water in sufficient quantity must be supplied to keep down the temperature.

It seems to be effective to circulate the water charged with SO_2, and, in the case of iron, with $FeSO_4$ and $FeCl_2$. But this must not be continued till the water becomes saturated with the H_2SO_4 and HCl in the case of charcoal, or with Fe_2Cl_6 in the case of iron. If this is done, the action seems to be deleterious, as the water loses its power to dissolve and decompose the gases SO_2 and Cl.

Charcoal and coke are cleaner than iron, as no sulphur is set free by their use. If a sufficient amount is used to give contact surface enough, they will, probably, prove as effective as iron. If charcoal is used, the greatest care must be taken in roasting it, to avoid a mechanical loss of gold. Coke has the advantage over charcoal in this respect, and also in furnishing less obstruction to the draft, and also in suffering less from crushing. Particular care must be taken that the water used does not get surcharged with sulphuric acid, or the efficiency of the condenser will be at once found to suffer.

Although iron will, perhaps, prove more expensive than charcoal or coke, the fact that a great quantity of waste iron accumulates about a mill will, perhaps, render its use possible. The value of scrap-iron to the chlorinator has already been established in precipitating the gold still left in the leaching waters after ferrous sulphate has apparently thrown down all the gold. In addition to the ce-

ment copper, for which this iron was originally used, enough gold is obtained to make a saving of several thousand dollars in the course of the year. If iron is used for condensation, it should be so placed as to offer the most surface to the gases, and to yield the least obstruction to the draft. It would, probably, be possible to obtain most of the gold by sluicing out the condenser from time to time and either settling out the gold or filtering it. It would not be necessary to remove the scrap-iron, or to dissolve it in sulphuric acid, as I had to do; it would only be necessary to add more scrap-iron at the top from time to time to supply the waste.

The chief difficulty that would have to be met in applying either method would be in constructing a condensing tower that would resist the action of the hot acid gases and water. I think, however, that this need not prove an insuperable obstacle, if the tower is built of ordinary brick laid in a mixture of dry sand and tar, instead of lime mortar, and this is lined with fire bricks of the best quality which have been boiled in a mixture of asphaltum and coal-tar of proper consistency so that the coating will not run when heated to the temperature of, say, boiling water. These bricks if laid while still hot will make tight joints without mortar, but if they are laid cold may be joined acid-tight by means of hot asphaltum of proper consistency. If properly built, such a tower ought to give very little trouble from acid waters. (Some excellent hints on this subject are given in the *Engineering and Mining Journal*, May 26, 1888, by Dr. Francis Wyatt, in his article on "The Development of the American Chemical Industry.") It will be necessary, however, not to use too much coal-tar in the mixture, or the lining will soften in spite of the water present and cause the gold, etc., to adhere to the walls, making a very disagreeable mass to clean up. Probably an experimental tower could be made of wood or an old leaching tank if it were well protected by asphaltum.

If charcoal or coke were used it would be necessary to provide means for charging these substances at the top, and for withdrawing them at the bottom, from time to time, when their gold-content was to be extracted.

It would also be necessary to provide an artifical draft to draw the fumes through the condenser.

A Guibal fan or a Root blower, constructed mainly of wood and thoroughly painted from time to time with asphaltum and coal tar, would serve the purpose here as it does in connection with quicksilver condensers, where they are equally exposed to the attack of

acid waters. The source of power (in northern California, at least, where water under a high head would most always be available), would most conveniently be a small Pelton wheel, and the additional expense incurred almost nominal. Where water-power was not at hand it might be found best to use the condenser only at such times as salt was added to the ore, as the volatilization loss at other times would be probably very small. It is, of course, evident that an abundant supply of SO_2 should be produced at such times as the salt was added on the finishing hearth, so as to aid in the condensation of the gold from the roasting ore on the upper hearths. The condensation waters that issue from the condenser should be filtered, after being allowed to settle, through a box containing charcoal so as to extract all suspended gold.

Whether or not such a condensing plant as I have sketched could be made to pay for itself on the working scale will depend on further experiments on a larger scale than any I have had the means of making. The results that I have obtained, however, are, it seems to me, sufficiently encouraging to warrant the attempt being made on not too extensive a scale at first, with such ores as are found to give too great a gold loss in roasting. The same appliances, somewhat modified, would probably serve as a valuable attachment to the dust chambers of silver roasting furnaces.

Conclusion on the Conditions that Reduce the Gold-Loss in Roasting to a Minimum.

In justice to the metallurgists of California, I should not neglect to add that the case of the excessive loss above cited is not to be taken as a type of the work that is done in treating such gold ores. It is, however, through just such expensive mistakes as the one instanced that the good results now common have been reached.

In April of this year (1888) I visited Nevada City for the purpose of learning the conditions of treatment that obtain at the present time. The only chlorination works then in operation at this place were those of Mr. Maltman and that at the Merrifield Mine, in charge of Mr. Stansfield. Both these works have favorable reputations. In addition there are other competing works at Grass Valley. While I have had no time or opportunity to determine for myself the exact yield at these works, the fact that they compete in the open market, charging from $15 to $17 for treatment according to size of lots, etc., and guarantee a yield of 90 per cent. of the gold

and 60 per cent. silver contents of these sulphurets, would seem to indicate that they do exceedingly good work in roasting, and that they have arrived at the conditions which reduce these losses to a minimum.

Here the old-fashioned long furnace with a single step separating the finishing hearth from the rest of the furnace is still the favorite furnace, the total lengths of the hearths being from 55 to 65 ft. by 10 ft. wide. The furnaces hold from 6 to 9 tons and usually produce about 3 tons per day, so that a lot of ore is in the furnace from two to three days at a time. The custom now is to charge the salt after a long oxidizing roast at a dull red and ending with a low cherry-red, where, as the ore reaches the finishing hearth, and the sulphur has been very thoroughly oxidized, the salt is added, the temperature of the furnace being slightly lowered at the time. The salt is stirred thoroughly into the ore, and as soon as it has become "dissolved" by the roasted ore, which takes about half an hour, it is drawn into the cooling pit.

After the experiments showing how the volatility of the gold in the presence of chlorine is increased by even a slight increase of temperature above a red heat, it will be readily understood how important an influence this lowering of the temperature, when the salt is added, has on reducing the gold-loss.

It must also be evident, that, since the ore remains in the furnace for from 48 to 72 hours, time is not the most important factor in causing the loss. Of course, if salt were present all this time the result would be probably somewhat different.

This brings me to a point that is well worth clear statement as I have never seen it mentioned before.* I have already shown, that, in all the muffle tests that I have made, I found that with a given percentage of salt the loss of gold was invariably less when the salt was mixed with the ore at the start, than when it was mixed with the ore after a long oxidizing roast.

I thought at first that this result was due to a mechanical loss caused by the decrepitation of the salt added to the hot ore; this cause was, however, carefully eliminated by fusing, and then pulverizing the salt and adding this dried salt carefully to the ore after it had slightly cooled. The result was the same. The real reason is apparent; it is that when the salt is added at the start, the chlo-

* Mr. Geo. J. Rockwell has recorded a similar result on the large scale with *silver ores* at Las Yedras, Mexico.—*Engineering and Mining Journal*, vol. xlv. p. 159.

rine is partly used in volatilizing the sulphur, and so adding a given amount of salt at the start, as far as the gold-loss is concerned, is practically equivalent to adding a less amount of salt after a long oxidizing roasting. Hence, whenever a batch of ore is treated at a time and a continuous process is not used, it is natural to expect a greater loss of gold when the salt is all added after a long oxidizing roast. But as it is well known that the reverse procedure gives the least loss of gold in continuous roasting on a large scale, as has been pointed out by Mr. Aaron, it is worth while to ascertain the reason why.

The reason of this difference, puzzled me for a long time; it is, however, undoubtedly due to the fact that in the muffle experiments, and in fact to all batch roasts in the reverberatory furnace or otherwise, where the whole lot of ore is kept at the same temperature throughout, when the gold chloride has once formed and left the batch of ore, that is the last of it as far as that sample of ore is concerned; hence the more chlorine set free after the gold becomes exposed to it the greater the loss. Hence adding salt at the end produces a greater loss of gold. Now, when the ore is treated continuously in the long hearth reverberatory furnace, it is not all kept at the same temperature, but it is red hot at one end and dark at the other; hence excellent conditions are maintained for condensing the gold chloride set free at the hot end by the cold ore still containing sulphur at the other. Now if salt is all added at the start there is a continued volatilization of chloride of gold all through the furnace, and a less favorable opportunity for it to condense, while if on the other hand the salt is all added at the hot end at the finishing hearth, although a rapid volatilization of gold takes place there, it has a chance to condense by coming in contact with the long surface (some thirty feet) of unsalted cooler ore; some of this is yielding SO_2 and with the steam from the burning fuel offers excellent means for reduction of the chloride of gold right within the furnace, but the most efficient means probably is the pyrites themselves.

In order to test the truth of this assumption, I passed a gold-charged chlorine stream over one-sixth assay ton of Murchie pyrite, which originally contained 0.75 mg. of gold. On gently warming the pyrite a most vigorous combination took place between the gold and the chlorine stream, brilliant luminous clouds of scale of a micaceous appearance filling the tube. These proved afterwards to be anhydrous chloride of iron; chloride of sulphur was afterwards formed; no gas escaped from the tube except when the chlorine stream was

urged, then the gold flame became apparent. The total amount of gold volatilized and sent to the pyrite was 5.19 mg. Much of this escaped condensation, for the ore after treatment (which by the way was a very uncomfortable mass of hygroscopic chlorides of iron, sulphur, etc.) assayed 3.30 mg. of gold. Now the original content of gold in the ore was only 0.75 mg., hence the gold-content of the ore had increased, owing to this cause, more than 300 per cent. Of course, this is an extreme case, but it shows how important a part the condensation of this volatilized chloride in the furnace itself may play in reducing the gold-loss. That there is something in this on a large scale is evidenced by the fact which I have always observed, that where the upper hearth was kept at a red heat there was a considerable loss of gold when salt was used, and that where the best results have been obtained the upper end of the hearth was kept below a red heat for a distance of fifteen or twenty feet.

I think enough has been said to show what a delicate process the chloridizing-roasting of gold ores is; how easy it is to incur enormous losses of gold by inattention to what may appear insignificant trifles; and how necessary in such matters is a systematic weighing and sampling, as well as assaying of the ores and products, in order to know at once when such losses are taking place, so as to be able at once to check them. The remarks on this latter head made in Mr. Daggett's paper on the Russell Process (*Trans.*, xvi. 363) are certainly worthy of careful consideration. To my certain knowledge, considerable losses have taken place through lack of attention to these simple matters of ordinary routine. Hap-hazard and occasional sampling and assaying are worse than useless. They lead to great losses of valuable capital, frequently to the total abandonment of good properties, and, worse than all, to a false sense of self-satisfaction that discourages improvement by denying its necessity.

THE COST OF A TON OF PIG-IRON IN THE SEQUACHEE VALLEY.

BY WILLIAM M. BOWRON, SOUTH PITTSBURGH, TENN.

(Birmingham Meeting, May, 1888.)

AN interesting calculation was made at the Chattanooga Meeting of 1885 as to the cost of making a ton of pig-iron in the Chatta-

nooga and Birmingham districts. Since that time new territory has been opened, new railroads have been built, and recent construction has remedied some of the leakages of former practice. The metallurgy of the ores of these districts and the capabilities of their fuels are now clearly ascertained; but the old question, How much does it cost to make a ton of iron? is still unanswered, so far as popular knowledge goes.

Cost-accounts are considered, in this district at least, as close secrets; and I am not prepared to betray them, for the excellent reason that I have never had access to the cost-account of any firm making iron in these districts.

Probably if I had sought such special information I could have got it, but it would not have been available for publication, and its possession would have rather been a source of embarrassment than aid in the independent investigation that I have made. Besides which, there are peculiarities in the conditions of most firms that prevent them from being representative of their neighbor's practice.

My first idea was to try to get in confidence such figures as might be averaged; but a very little study made me abandon this, if for no other reason than that the differences in matters that were included in cost would render such average worthless. For example, one operator builds fifty coke-ovens and charges them up to "general expenditure." His neighbor, building the same number, charges them up to "capital account." The figures of cost on the same make and under similar conditions would not be identical. I now propose to give some data of cost that may assist those making their own calculations for any specific locality, premising that my figures are based on Sequachee Valley practice, as the district most familiar to me. It is but simple justice to the Tennessee Coal, Iron and Railroad Company to state that none of the figures have been derived from their work. Owing to their special facilities they work one department into another, and the figures I arrive at should not be quoted against them. I am dealing with a furnace built in Sequachee Valley to work its own local ore and coal and to buy its soft ore.

The materials included in Sequachee Valley are ore and coke.

Hard ore is worth 75 cents per ton and soft ore can be had for about $2.25 per ton delivered.

A working mixture is,

Hard ore, ...	4200 pounds.
Soft ore, ...	3500 pounds.

Allowing the hard ore to run 30 per cent. and the soft ore 50 per cent., this charge would give 3010 pounds of iron. Reducing the ore to that required for 2000 pounds of iron, we have

Hard, 2757 pounds, worth,	$1.034
Soft, 2325 pounds, worth,	2.615
Add 10 per cent. for waste, moisture, etc.,	.365
Ore per ton of iron,	$4.014

The next item is coke. Analyzing its probable cost, I have from different sources and composite data got the figures below.

The general nature of the coal may be gained from the following table:

NAME OF COAL.	Miles North of S. Pittsburg.	Ash.	Fixed Carbon.	Volatile Matter.	Sulphur in Ash.
1 Battle Creek	0	9.28	73.18	17.54	1.28
2 Wall No. 2	2	11.03	72.65	16.32	1.37
3 Wall No. 3	2	12.14	69.64	18.18	1.00
4 Wall No. 5	2	9.16	70.52	20.32	0.23
5 Wall No. 7	2	7.29	64.57	28.14	0.65
6 Ralston's Cove*	3	14.13	65.65	20.22	0.90
7 Whitewell	18	11.17	60.62	28.21	0.12
8 Victoria	14	10.40	65.00	24.60	1.86
9 Bee Branch	16	13.77	60.00	26.23	0.63
10 Distribute Branch	18	12.07	61.00	26.18	0.50
11 Hammet's Cove	22	10.14	60.59	29.27	0.66
12 Griffith's Creek	22	9.63	59.32	31.05	0.81
13 Kinnaird's Cove*	28	10.02	60.54	29.44	0.33
14 Deakin's*	30	11.16	60.84	28.00	0.42
15 Stone	31	10.14	60.14	28.72	0.60
16 Rankin	31	14.33	61.39	24.28	0.81
17 Elliot	33	9.21	60.47	30 33	0.39
18 Heard	35	8.27	65.37	28.36	0.89
19 Seals†	40	5.62	73.64	20.00	0.84
20 Norwood†	45	8.83	73.28	16.13	1.76
21 Furguson†	55	9.36	71.01	17.86	?
22 Webb	70	13.98	65.40	20.62	1.48
23 Panter	44	8.84	69.77	21.39	1.40
24 Basse Creek	73	12.38	58.83	28.78	0.08
25 Brown	75	11.45	55.87	32.06	0.11

I have made an attempt to divide up the cost of mining, as follows:

* Outcrops. † Dickerson.

	Cents.
Mining coal,	50.0
Air course and entry,	12.5
Incline,	3.0
Superintendence, clerks and offices,	2.5
Mules, drivers, and outside labor,	6.8
Tipple,	2.5
General expenses, *i.e.*, taxes, insurance, exhaustion of land, etc.,	7.5
Timber,	2.5
	87.3

On cars at mine. Coked in 11-foot ovens, holding 4 tons of coal, 100 bushels of this coal give 115 of coke; or 8000 pounds of coal give 4600 coke. The cost of this may be divided thus:

	Cents.
4 tons coal per oven,	3.492
Charging, levelling, bricking and drawing,	.500
Loading,	.250
Repairs,	.050
Extra labor, switching, weighing, etc.,	.150
	4.442

Or $1.929 per ton of coke on cars at ovens.

To make a ton of pig-iron with this coke takes 2748 pounds of coke, worth

Net cost,	$1.929
Waste and braise, 10 per cent.,	.193
	$2.122

Or $2.915 per ton iron, plus the freight for haulage from coal-mines to furnace.

To recapitulate:

Ore,	$4.014
Coke,	2.915

Labor.

We can only arrive at labor by considering the actual production of furnaces using similar materials to those under consideration.

The following wages and labor are taken from actual practice:

2 Engineers, at	$1.90	$3.80
2 Firemen, at	1.25	2.50
2 Keepers, at	2.50	5.00
2 First helpers, at	1.15	2.30
2 Second helpers, at	1.10	2.20
2 Third helpers, at	1.00	2.00
2 Stovemen, at	1.10	2.20
1 Blacksmith, at	2.00	2.00
1 Helper, at	1.15	1.15
1 Lampman, at	1.10	1.10
5 Water boys, at	.40	2.00
4 Cindermen, at	1.15	4.60
1 Foreman cinder-yard, at	1.25	1.25
14 Laborers, at	1.15	16.10
4 Horses, at	1.00	4.00
1 Scrap-man, at	1.25	1.25
1 Scales-man, at	1.40	1.40
1 Scales-man, at	1.30	1.30
2 Cage-men, at	1.15	2.30
2 Top-fillers, at	1.20	2.40
2 Top fillers, at	1.15	2.30
9 Ore-fillers, at	1.00	9.00
8 Coke-fillers, at	1.00	8.00
4 Ore-breakers, at	1.00	4.00
4 Ash-men, at	1.00	4.00
2 Water-boys, at	.40	.80
1 Foreman,	1.45	1.45
25½ Laborers on ore in stock-house, at	1.00	25.50
		$115.90
Switch-engine and yard crew,		20.00
Superintendence and clerks,		20.00
		155.90

Or an average make of 85 tons per day, or $1.834 per ton. We have then,

Ore,	$4.014
Coke,	2.915
Labor,	1.834
Stores,	.250
(This includes railroad-iron, oil, coke-forks, sand, lumber, etc.)	
	$9.013

Being the cost of making a ton of pig-iron *less* the cost of bringing the fuel to the furnace, which is supposed to be located near the mine in the Sequachee Valley, and 25 cents should cover it. There only remains to add for repairs and depreciation of plant 10 per

cent. on $100,000 investment and 6 per cent. interest on the same for use of the money (for the only safe way is to regard the money invested as borrowed). These, calculated on 30,000 tons per annum, are,

Depreciation, ...	.25
Interest, ...	.15
Brought forward, ...	9.013
	$9.413
Probable freight on coke, ...	.25
	$9.663

As this is a ton of 2000 lbs. the cost of a ton of 2240 lbs. will be, according to the above figures, $10.82.

The allowance for sand in the pig-iron ton does not require to be made here.

Local conditions vary, but the figures above will come very near the truth in Sequachee Valley, where ore and coke are only 4 miles apart in a direct line, and can commercially be united by rail inside 20 miles.

As a basis for comparison this estimate will be useful if only to check the wildly small estimates of the authors of "boom" literature and their residuary legatees, the tariff-tinkers. Unless conditions are favorable, construction suitable, and management good, these figures will be exceeded. Distance from market becomes a further factor in the question of "profit and loss," but I am simply regarding here the cost of making a ton of iron in Sequachee Valley.

THE EFFICIENCY OF A STEAM-BOILER USING THE WASTE GAS OF A BLAST-FURNACE AS FUEL.

BY D. S. JACOBUS, M.E.

PRESENTED BY J. E. DENTON, HOBOKEN, N. J.

(Birmingham Meeting, May, 1888.)

THE boiler here referred to was of the water-tube type, having 2535 square feet of heating-surface, which the makers held to be capable of generating 325 horse-power of steam; this being understood to mean that 325 times 30 pounds of water would be evap-

orated per hour from feed-water, at 212° F., into steam at atmospheric pressure.

The blast-furnace gases were led by an underground conduit to a point in front of the firing-door of the boiler, and thence vertically upwards to a chamber which was built out from the firing-door opening for about 2 feet. This projecting chamber was about 20 by 16 inches in cross section, and its interior was divided into eight horizontal gas-passages, 3 inches deep and 7½ inches wide. The air-passages, six in number, were 2 inches deep and 7½ inches wide, and were arranged alternately with the gas-passages, an air-passage being between each two gas-passages. Each gas-passage connected with a space at the outer extremity of the chamber, and into this space the vertical gas-flue delivered the hot waste gas. Each of the air-passages was open at either side of the chamber to the atmosphere, at which point regulating-valves were placed. By means of the draught of the boiler-chimney, air and hot gas were drawn through the horizontal passages of the chamber, and brought into contact with each other at the sill of the firing-door opening, where the mixture was ignited by passing over a bed of incandescent coal, which was maintained on the grate of the boiler for this purpose.

Two tests of the boiler were made, each over an interval of 12 hours. The water fed and evaporated was measured through a Worthington meter, arranged so that 20 cubic feet of water could, at intervals, be drawn off through it upon a pair of scales, and thereby the rate of the meter determined under the exact conditions of its service. The correction-constants at the foot of Tables I. and II. show the results of such tests of the meter.

The composition of the gas entering and leaving the boiler-furnace was determined simultaneously by the use of two portable Elliott gas-analyzing instruments; Table III. exhibits the data thus obtained.

The temperatures of the entering gas, of the combustion-chamber or space over the grate, and of the gases at their exit from the boiler-furnace, were respectively determined by a copper-ball pyrometer. Owing to losses by radiation, while removing the ball from the furnace, the temperatures obtained are lower than the true amounts; but as the only temperature equal to a red heat is not employed in deducing any of the conclusions made herein, the amount of error is regarded as unimportant. Columns 10 and 11 of Table II. give the results obtained.

The quantity of air supplied to the furnace was approximately

measured by the use of two anemometers, or current-meters, placed respectively in a circular sheet-iron pipe 9 inches diameter, which was made to embrace one-half of the openings for the inlet of air in the projecting chambers, through which air and gas were introduced into the boiler-furnace, as described above. Columns 3 and 4 of Table II. give the readings of the anemometers in linear feet, which, being multiplied by four times the area of a 9-inch circular pipe, expressed in square feet, and by the correcting multipliers in column 16, give the theoretical volume of air in cubic feet. The theoretical amount of air thus determined is increased by 20 per cent. in the formulæ of Appendix I., but, as there stated, the record of the anemometers is not regarded as correct. The unreliability of the anemometer-record is shown as follows: From the analyses of the gases, the ratio of the weight of the entering gas to the weight of air is determined (see Appendix I.). If, then, the weight of air as shown by the use of the anemometer, is considered to be all the air that enters the furnace, and the weight of gas consumed is calculated by multiplying this weight by the ratio of gas to air, we obtain a weight of gas, the calorific effect of which, as calculated in Appendix II., will not be great enough to produce the known amount of heat required to evaporate the water fed, and leave a reasonable amount for draught and radiation. The amount of air as shown by the anemometer is, therefore, too small. The pressure of the entering air just within the air-orifices of the mixing chamber, and that of the entering gas, are given in columns 7 and 9, Tables I. and II.; the pressure at the base of the chimney causing the draught is given in column 8. All these pressures are in inches of water. A minus sign opposite these figures indicates pressure below the atmosphere.

The fluctuation in the signs and magnitude of the pressure of the entering gas, is due to the fact that the amount of gas available for generating steam was limited to the surplus of gas over the amount necessary for the hot-blast stoves; and this surplus was, at intervals, so exhausted, that the draught of the stoves reduced the pressure in the main waste-gas pipes leading from the blast-furnace, to less than that of the atmosphere, to such an extent as to change the sign of the pressure of the gas at its entrance to the boiler-furnace. The fuel used in the blast-furnaces was Walston coke.

The quality of the steam, or amount of priming, was determined by the use of a surface-calorimeter, the range of accidental varia-

tion of which was known to be not more than 3 per cent. of moisture. The results of these tests are given in column 10, Table I.

Opportunity offering, analyses of the gases were made at the entrance and exit of one of the Whitwell hot-blast stoves with the results shown in Table IV.

Conclusions.

The principal practical results deduced from the tests are as follows:

I. It is evident from the analyses given in Table III., that the only combustible element in the gas was carbon, and that the combustion was not entirely perfect, 1.8 per cent. of carbonic oxide being present when the gases reached the chimney.

II. Including the effect of the 650° initial temperature, the calorific power of a pound of the gases burned under the steam-boiler was 1413 British thermal units. For the detailed calculations by which this figure is deduced see Appendix II.

III. The amount of air required for the combustion of each pound of this gas was $\frac{9}{10}$ of a pound. For details of the method by which this result was deduced see Appendix I.

IV. With a chimney-draught equivalent to 1⅛ inches of water, an area of gas-inlet equal to 300 square inches, and an area of air-inlet equal to about 100 square inches, the boiler generated about 336 horse-power, on the basis of 30 pounds of water evaporated per hour, from a temperature of 200° feed-water, and at about 70 pound steam-pressure; the priming tests, Table I., showing that practically dry steam was produced. This is equivalent to the evaporation of 11.592 pounds of water per hour from and at 212°, or about 5 pounds per hour per square foot of heating-surface, which is as great a rate of evaporation as is obtained by firing with good coal at a rate of combustion of about 12 pounds of coal per square foot of grate-area per hour.

V. The efficiency of the boiler is 61 per cent., that is, of the 1413 thermal units or calorific power of each pound of the blast-furnace gas, 61 per cent. is probably represented by the heat usefully applied in evaporating the water fed to the boiler. This efficiency is equivalent to the evaporation of 8 pounds of water per pound of coal from and at 212°, assuming that the coal yields 12½ per cent. of ashes. For the detailed calculation by which the efficiency is deduced see Appendix II.

Appendix I.

From the composition of the gases entering and leaving the boiler-furnace, as shown by Table III, we find that the theoretical amount of air necessary to supply the oxygen for combustion, was 0.9 of a pound, per pound of entering-gas. This is arrived at as follows:

Let C_g = weight of carbon in 1 pound of entering-gas.
Let C_f = weight of carbon in 1 pound of flue-gas.
Let C_c = weight of carbon in the ignition-coal consumed per pound of total air used.
Let W = weight of air required for w pounds of entering-gas.
Let w = weight of entering-gas per unit of time.

Then we must have

$$C_f\ (w + W + W\,C_c) = w\,C_g + W\,C_c$$

Whence,

$$\frac{W}{w} = \frac{C_g - C_f}{C_f\ (1 + C_c) - C_c}$$

We have from the data in the tables

$C_g = 14.37$ per cent.
$C_f = 7.95$ per cent.

Coal used per hour for igniting gas = 71 pounds. Air recorded per hour by anemometers + 20 per cent. for leakage through openings in boiler-setting = 8459 pounds. Assuming the coal to be .80 per cent. carbon, we have

$$C_c = \frac{0.8 \times 71}{8459} = .0067$$

Hence

$$\frac{W}{w} = \frac{.1437 - .0795}{.0795\ (1 + .0067) - .0067} = .88$$

It will be noticed that this computation depends upon the carbon ingredient of gases alone.

A computation based upon the oxygen ingredient gives $\frac{W}{w} = .96$.

It may also be noticed that the effect of the amount of air obtained from the anemometers enters the computation.

We do not regard the anemometer-record as a reliable measure of the amount of air used; and our arbitrary allowance of 20 per cent. for leakage, which is based upon a rough estimate of the amount of leakage-openings noted by the observer, adds to the approximate

character of the figures used for air-consumption per hour. But if the value of C_o in the formula were made zero, that is, if the amount of air used per hour were made infinity, the resulting value of $\frac{W}{w}$ would be 0.8, a figure differing but 0.08 from that obtained with the value of air assumed.

Hence it is evident that the value of $\frac{W}{w}$ may be taken at the average of the above figures, which we have called 0.9, without apprehension regarding an impracticably erroneous result, due to the approximate character of the anemometer-record.

Appendix II.

The combustion of each pound of entering-gas is accompanied by the following development of calorific effect:

First. The 26.7 per cent. of carbonic oxide in burning to carbonic acid develops $.267 \times 4400 = 1175$ British thermal units.

Second. The pound of gas enters the furnace at a temperature of 584° F. above the temperature of the boiler-room. Therefore, taking the specific heat of the gas at .24, the available heat from this source is $584° \times .24 = 140$ British thermal units.

Third. There is consumed an amount of coal for ignition-purposes equal to

$$\frac{71}{8459 \times \frac{1}{9}} = .0075 \text{ pounds}$$

per pound of entering gas, from which there is available, assuming 12.5 per cent. ashes, an amount of heat equal to $.0075 \times 13000 = 98$ thermal units. The total heat available per pound of entering-gas is, therefore, $1175 + 140 + 98 = 1413$ thermal units.

There is wasted out of this amount:

1*st.* The heat necessary to raise the temperature of 0.9 of a pound of air from the temperature of the boiler-room, 66° F., to 775° F., the temperature of the chimney, making $(775° - 66°) \times .24 \times 0.9 = 153$ thermal units.

2*d.* A portion equal to the heat necessary to raise the gas itself from the boiler-room temperature to the chimney temperature, making $(775° - 66°) \times .24 = 170$ British thermal units.

3*d.* The heat necessary to raise the gaseous products of the igniting-coal from the temperature of the boiler-room to that of the chimney. For 12.5 per cent. ashes, the weight so raised will be, per pound of

entering-gas, $\frac{1}{8} \times .0075$ pounds, and the loss of heat will, therefore, be $(775° - 66°) \times .24 \times \frac{1}{8} \times .0075 = 1.1$ thermal units.

4th. The loss of heating-effect due to unconsumed carbonic oxide.

The analysis of gases at exit from furnaces shows that the escaping gases contained 1.8 per cent. of unconsumed carbonic oxide. There is 0.9 of a pound of air admitted per pound of entering-gas, and 26.7 per cent. of the latter is carbonic oxide.

Hence the total carbonic oxide is

$$\frac{26.7}{1.9} = 14.0 \text{ per cent.}$$

of the total gas escaping from the furnace.

Hence 1.8 per cent. of unconsumed carbonic oxide per pound of escaping gas is equivalent to a proportion of the total amount of carbonic oxide being unconsumed equal to 1.8 per cent. divided by 14.0 per cent., or 12.9 per cent.

TABLE I.

Test No. 1 on Boiler in Battery at extreme left of Boiler Room. September 24th, 1887.

Area of Gas-inlets, 340 sq. in.—Area of Air-inlets, 120 sq. in. Heating-surface of Boiler, 2535 sq. ft.

Time.	Water Meter.	Steam Pressure, lbs.	Temperature of Feed. Deg. F.	Percentage of Moisture in Steam.	Draught in Inches of Water.			Remarks and Total Results.
					Height of Water in Boiler Glass.	At Entrance to Boiler Furnace.	At Entrance to Chimney.	Pressure of Entering Gases.	
					Inches of Water.				
10.00 A.M.	211,728	65	198		5	—.5	—1.25	—.125	Temp. Boiler Room = 58°.
11.00 "	211,929	91	178	1.5	7.5	—.625	—1.25	— .25	Total Water Evaporated from and at 212° = 130,200 lbs. in 11½ hours.
12.00 M.	212,110	70	194		5	—.625	—1.375	—.125	
4.00 P.M.	212,793	76	158		7	—.75	—1.50	—.125	
5.00 "	212,952	59	135		7	—.375	—1.375	—.125	H. P. Centennial Standard = 328.2 horse-power.
6.00 "	213,120	57	182		6	—.50	—1.25	— .25	
7.00 "	213,275	54	185	1.1		—.625	—1.125	— .19	At 30 lbs. from and at 212°, Basis upon which maker's guarantee was made. 377 H.P.
8.00 "	213,460	62	184			—.50	—1.25	0	
9.00 "	213,651	62	146	.4	9				
9.30 "	213,654	65	146		5				Amount of Air not Measured.
Sum,	cu. ft. 1991.58*	661	1706	3.0	51.5	—4.5	—10.375	—1.19	
Average,	Per hour 173.2	66.1	170.6	1.0	6.4	—.56	—1.30	— .15	

* Corrected.

TABLE II.

Boiler Test No. 2, of Boiler at Extreme Left of Boiler Room, September 28th.

Time.	Water Meter.	Left Anemometer, K. & E.	Right Anemometer, Institute's.	Steam Pressure, lbs.	Temperature of Feed. Deg. F.	Draught. Pressure at Entrance to Boiler Furnace.	Draught. Pressure at Entrance to Chimney.	Pressure of Gas at Entrance to Furnace.	Temperature in Deg. F. of Entering-Gas.	Temperature in Deg. F. of Flue-Gas.	Temperature in Deg. F. of Air in Boiler Room.	Height of Water in Glass. Inches.	Area in sq. in. of Gas Inlet.	Area in sq. in. of Air Inlet.	Remarks and Total Results.
		Velocity of Air entering Furnace, in linear feet.				Inches of Water.									
10.00 A.M.	0214035	0057700	7121400	48	120	—.75	—1.25	+.5			67	5.0	300	96	H. P. Centennial Standard = 344.4 H. P.
11.00 "	0214228	0100900	7183500	60	184	—.75	—1.375	0			68	4.5	"	"	
12.00 M.	0214315	0143800	7243350	52	144	—.375	—1.5	+.375			70	1.0	"	"	H. P. at 30 lbs. per hour from and at 212° = 396.1 H. P.
1.00 P.M.	0214545	0181900	7298500	54	182	—.50	—1.5	+.875			70	2.0	"	"	
2.00 "	0214770	0229100	7367000	49	138	—.50	—1.25	—.125			70	5.0	"	"	
3.00 "	0214922	0276800	7535500	58	158	—.75	—1.25	—.25			70	6.0	"	"	Pounds of water evaporated per hour = 10924 lbs.
4.00 "	0215029	0323400	7504000	51	165	—.625	—1.375	—.25	550		68	6.0	"	"	
5.00 "	0215480	0366900	7566000	54	182	—.50	—1.375	+1.25			65	4.0	"	"	From and at 212° evaporated per hour 11883 lbs.
6.00 "	0215438	0406600	7625100	48	140	—.625	—1.375	—.25			64	7.0	"	"	
7.00 "	0215510	0453600	7694400	90	162	—.75	—1.75	+.75	750	850	63	0.0	"	"	Velocity of air entering furnace, measured through 9″ pipe covering one-half total air openings to furnace for each anemometer.
8.00 "	0215770	0498100	7749900	80	152	—.50	—1.25	0.0			62	9.0	"	"	
9.00 "	0215895	0545300	7807000	98	150	—.50	—1.375	+.5		700	62	3.0	"	"	
10.00 "	0216100	0585500	7859900	55	170	—.50	—1.25	+1.25			61	5.0	"	"	
Sum.	cu. ft. 2147.6*	527800	738500	797	2047	—7.625	—17.87	4.625	1300	1550	860	57.5	"	"	Multiple for K. & E. anemometer, 1.07; for Institute's, 0.83.
Average.	178.97*	47062*	51080*	61.3	157.5	—.587	—1.375	+.356	650	775	66.2	4.4	300	96	Temp. of Combustion Chambers, 1600° F.

* Corrected.

Correction for meter: 1.04 × actual reading of meter = true cubic feet.

Consequently the heat wasted per pound of entering-gas, due to unconsumed carbonic oxide is $.129 \times 1175 = 152$ thermal units.

5th. The heat equivalent to the radiation taken at, say 5 per cent.* of the total heat of combustion, or $.05 \times 1413 = 71$ thermal units.

The total loss of heat is therefore $(153 + 170 + 152 + 71 + 1.1) = 547$ thermal units.

The efficiency of the boiler would therefore be equal to

$$\frac{1413 - 547}{1413} = 0.61$$

This would indicate a useful effect, equivalent to the evaporation of $.61 \times 15 = 9.15$ pounds of water from and at 212° F. per pound of combustible, or, if coal had been used as fuel, an evaporation of $9.15 \times \frac{7}{8} = 8$ pounds of water per pound of coal, if the ashes and refuse amounted to 12.5 per cent. of the amount of coal fed to the furnace.

Test of meter under actual conditions of pressure and temperature of feed-water, determined that 1.03405 times actual reading of meter gives true cubic feet.

TABLE III.

Analyses of Gases made during Boiler Tests, September 28th.

Time Sample was Taken.	At Entrance to Furnace. Per cent. by Volumes.				At Exit from Furnace and Entrance to Chimney. Per cent. by Volumes.			
	CO_2	O.	CO.	Nitrogen and Hydrocarbons.	CO_2	O.	CO.	Nitrogen and Hydrocarbons.
10.30 A.M.	6.9	0	28.8	64.3				
11.30 "					11.4	9.0	0.0	79.6
5.20 P.M.	7.7	0	20.1	72.2	22.0	0.8	3.3	73.9
6.45 "	6.9	0	28.7	64.4	21.3	0.7	2.9	75.1
8.30 "	6.6	0	29.7	63.7	18.7	3.4	.0	77.9
9.45 "	7.3	.5	31.7	60.5	19.8	.9	3.7	75.6
Average	7.08	.10	27.80	65.02	18.64	2.96	1.98	76.42

* See Report on Trial of Boilers at Pacific Mills, by J. C. Hoadley, Vol. VI. *Transactions* American Society Mechanical Engineers.

TABLE IV.
Whitwell Hot-Blast Stove.

Time Sample was taken.	Approximate Distance Gas Valve was Opened. Inches.	Area of Opening of Gas Valve. Sq. in.	Average Distance Air Valves to Combustion Chamber were Open. Inches.	Area of Opening of Air Valves. Sq. in.	No. of Air Valves Open.	Draught at Chimney Flue in inches of water.	Time Blast was shut off and Gas made to Burn in Stove.	Composition of Gas per cent. by Volume. At Entrance. CO_2	At Entrance. O	At Entrance. CO.	At Entrance. Nitrogen and Hydrocarbons.	At Exit. CO_2	At Exit. O	At Exit. CO.	At Exit. Nitrogen and Hydrocarbons.	Calculated ratio of Air per lb. of Gas from per cent. of Carbon shown by Analysis.	
5.15 P. M. Sept. 27	8	124	1.5	150	2		5 P. M.	8.2	.8	28.7	62.3	16.5	3.6	0.0	79.9		Samples of Gas analyzed *not* taken simultaneously from entrance and exit respectively.
5.50 " " 27																	
10.00 " " 27							9.45 P. M.	13.3	.5	22.6	63.6	20.4	3.1	0.0	76.5	1.07	
10.45 " " 27								Avge. 10.75	.65	25.65	62.95	18.45	3.35	0.0	78.2		
4.25 " " 29	6	84	.94	94	2	1.625	4.10 P. M.	6.2	.5	18.1	75.2	16.1	2.7	0.0	81.2		Samples of Gas analyzed taken simultaneously from entrance and exit respectively.
5.00 " " 29																	
5.30 " " 29	6	84	1.19	119	2	1.625	5.20 P. M.	12.2	.1	22.5	65.2	15.1	6.3	0.0	78.6	.97	
5.30 " " 29								Avge. 9.2	.3	20.3	70.2	15.6	4.5	0.0	79.9		

From the average of the analyses we have the following per cent. by weight:

	At Entrance.	At Exit.
CO_2.....	10.69	26.37
O	.11	3.05
CO.....	26.71	1.78
Nitrogen.....	62.48	68.80
C in CO_2.....	2.92	7.19
C in CO.....	11.45	.76
Total C.....	14.37	7.95

Ratio by weight of gas at entrance to air used, calculated from decrease in per cent. of the total carbon and the oxygen as shown by analysis $= 1.11 = \frac{\text{Gas}}{\text{Air}}$.

HENDERSON STEEL.

BY ALFRED F. BRAINERD, BIRMINGHAM, ALA.

(Birmingham Meeting, May, 1888.)

THERE has been no enterprise undertaken in this and adjoining States which has attracted so much interest, or has been watched so closely as this, the first successful attempt to convert our ordinary high-phosphoric pig-iron into steel from native material.

Our iron-masters have noticed intently every attempt to convert high-phosphorus pig-irons into steel by the Clapp-Griffith process in this country, and the basic process on the Continent, and studied the various patents aiming to accomplish the same purpose, and the various litigations arising among various patentees over their rival claims; but up to August, 1887, nothing was done, and no move had been made to introduce any process for making steel.

At the time above named, Mr. James Henderson, of New York, came to this place with a view of introducing his patent furnace and other metallurgical processes and appliances. After some delay and personal interviews between parties interested and the inventor, a party of gentlemen, all of moderate means, and entirely unacquainted with the manufacture of either steel or iron, decided to form themselves into a body corporate, known as the Henderson Steel and Manufacturing Company, with a capital stock of $20,000,

half of which was supposed to be sufficient to demonstrate the utility of the Henderson process.

An old plant, located at Boston, Massachusetts, which had been built for the same purpose several years before, was removed to the furnace-site located at North Birmingham, Alabama, within a pistol-shot of the North Birmingham furnaces, a part of the Sloss Steel and Iron Company's plant, and on a side-track of the Georgia Pacific Railroad; also near the Bessemer branch of the Mineral Railroad, of the Louisville and Nashville system, and also at the intersection of Village Creek and the North Birmingham dummy-line, and about half way from the city of Birmingham (about three miles distant) to North Birmingham. There is a ledge of dolomite or magnesian limestone running through and cropping out on the Henderson Steel and Manufacturing Company's property which has been used in building their plant and also in the metallurgical operation of the furnace. This stone was to have been used in the running of the North Birmingham furnace and by the original projectors of that furnace. The analysis is given below. The capacity of the plant is about one and one-half tons per heat, and it was originally intended to make some ten to twenty heats a day. Owing to an unfortunate gas-explosion at the first melt, the furnace was so badly shaken up that it has only been run spasmodically, making some three to four melts during the day, and consequently doing nothing like its duty or estimated capacity.

This plant consists of a gas-producer connected with two small engines, and by them supplied with air through several tuyeres. This producer is fitted up with a bell and hopper for charging the coal or any flux necessary for slagging off the ash into cinder, and also with a tap-hole and cinder-notch. It was the intention of the inventor to use in the producer coal with a large percentage of volatile matter and of non-coking quality, or to use a "slack-coal," or a coking coal with slate and other impurities which would be a cheap fuel. This idea was finally abandoned, and the best grade of Cahaba coal was used (analyses given below.)

The producer is connected with the furnace, which is a small open-hearth, with revolving bottom and sand-joint, and connects with a heating and melting furnace, serving the purpose of a cupola furnace; then this is connected with the masonry and boilers, thence into a stack, etc. There is also a condenser to condense gases and fumes, which answers as a sort of dust-chamber. There are also a

rotary engine and very large rotary crusher used in preparing patent bricks and bottom for furnace.

In operating the furnace, after drying out and heating up with ordinary coal, the bell is let down, and coal is discharged into the hoppers; from them it is fed down and equally distributed into the producer, where it is ignited and burnt. Air in certain proportions is merged with the gas after combustion has taken place at the tuyeres, and thence it passes along through the open-hearth furnace over to the melting and heating-furnace, where a lot of pig-iron is placed. Thence the gas passes under the boilers, thence into the condensing-chamber, or directly into the stack.

The iron being melted, the open-hearth furnace is revolved around until an inlet opening comes opposite the tapping-hole of the melting-furnace, and the charge is withdrawn and run into the open-hearth furnace, which is revolved around to proper position, and then the process of dephosphorizing, desulphurizing and decarbonizing goes on. When, after taking out a test-ingot, the carbon has been found to be reduced, say from 3 per cent. to 0.12 per cent., ferro-manganese or spiegel is charged, and allowed to mix and incorporate itself with the mass of molten steel. When this has taken place the furnace is tapped, and the steel runs off into a ladle, whence it is cast through an orifice in the bottom into ingot-moulds, placed already in the casting-pit during the pouring; and during the converting process a fresh lot of iron is melted in the heating-furnace. When another melt is ready, the above process is repeated, and so on, day and night. It is interesting to note, by comparison of the analyses given below, the changes in converting some very poor white and mottled iron into steel, both hard and mild, or soft steel. This steel has been tested physically, and has a tensile strength of 85,000 pounds per square inch. It has been successfully made into hammers, cold-chisels, jack-knives, razors, clocks and watch-springs, paper-knives and medals.

It was the aim of the inventor to use only the poorest grades of pig-iron obtainable; and after a great deal of trouble some poor enough to suit his fastidious taste was found, and cost $10 per ton free on board at furnace.

The Company has decided to increase its capital stock and build* some larger furnaces.

It is not claimed that this furnace is destined to supersede the

* A five-ton furnace is now (August, 1888) in process of erection, the old furnace having been torn down July 2d, 1888.

Bessemer or the basic process, but it is especially adapted to our ores and pig-irons. Its relative cost we have had no means of ascertaining, and it is not within the province of this paper to discuss, but it is believed that it can be made to pay a profit to the producer. Certainly the experiments so far have been successful in showing what can be done with Alabama pig-iron.

The following are the analyses of the raw materials and products of the furnace:

Dolomite or Magnesian Limestone.

Protoxide of iron,	0.3100
Silica,	4.7500
Lime carbonate,	62.4833
Magnesia carbonate,	32.4421
Sulphuric acid,	0.0065
Phosphoric acid,	0.0078
	100.0000
Sulphur,	0.00274
Phosphorus,	0.00326

Hard Red Ore from Ft. Payne L. and Imp. Co., Ft. Payne, Ala.
(Used as a reducing agent in the furnace.)

Peroxide of iron,	51.770
Silica,	4.690
Alumina,	1.288
Lime,	22.909
Magnesia,	0.122
Binoxide of manganese,	0.270
Sulphuric acid,	0.250
Phosphoric acid,	0 831
Carbonic acid,	18.060
Undetermined,	0.081
	100.000
Iron,	36.239
Sulphur,	0.1000
Phosphorus,	0.3631

*Cahaba Coal.**

Moisture,	2.24
Volatile matter,	34.12
Fixed carbon,	60.75
Ash,	2.41
Sulphur,	.48
	100.00

* This analysis is by J. B. Porter & Co., Cincinnati; the rest are by the writer.

White Iron.

Carbon,	2.9778
Silicon,	0.9605
Phosphorus,	0.5781
Sulphur,	0.3286

Mottled Iron.

Carbon,	2.9626
Silicon,	1.9597
Phosphorus,	0.6493
Sulphur,	0.2246

No. 1 Henderson Steel.

Carbon,	0.750
Silicon,	0.0093
Phosphorus,	0.0513
Manganese,	Trace.

No. 2 Henderson Steel.

Carbon,	0.200
Phosphorus,	0.0757
Manganese,	0.7700

DISCUSSION.

A MEMBER: It seems to me that the most important part of this paper is the statement "Then the process of dephosphorizing, etc., goes on." I should like to know what it is that goes on.

MR. BRAINERD: I do not profess to understand the theory of the process, nor did I intend to state or criticize it in the paper, which is merely intended to put on record the analysis of the materials, which I made myself, and for which I can vouch. I know that out of these pig-irons these steels were made.

NOTE BY THE SECRETARY: After some further desultory discussion, the President of the Henderson Steel Co. invited members to visit and examine the furnace for themselves; and during the Birmingham meeting, a considerable number availed themselves of this invitation; but no opportunity presented itself for a renewal of the discussion. As the incomplete statement given in the foregoing paper appeared to me to be unsatisfactory to Mr. Henderson, I invited him to furnish a brief account of the apparatus and process, which he has done as follows:

"The Henderson steel-making apparatus consists of a gas-producer, a steel-refining chamber, a purifying-chamber, an air-heating and boiler-furnace chamber, a boiler, and an ammonia refrigerating-chamber (which is to be added) with two small blowing-engines.

"The gas-producer is a cupola, lined with fire-brick, with air-tuyeres around the hearth, and a retort at the top, closed with a bell-and-hopper. The outlet-neck of the gas-producer is the inlet of the steel refining-chamber, which immediately adjoins it; the outlet-flue of the refining-chamber is the inlet-flue of the purifying-chamber, the outlet-flue of which leads to the air-heating and boiler-chamber, where the heat of the spent gases is mostly taken up in heating the air-blast for burning the gases and raising steam.

"In working the apparatus, the gas-producer is always kept full to the top with coal (and flux to scorify its ash). Blast is applied through the tuyeres, and the heat caused thereby produces destructive distillation of the coal in the retort above, so that by the time the fuel reaches the bottom of the retort all of its volatile matters are given off and it has become coke, which passes to the tuyeres, where it meets the air, forced in by one of the piston-blowers in exact measured quantity, and converts it to carbonic oxide, while at the same time the flux keeps the fuel free of ash, so that a clean surface of coke is always exposed to the air. This permits the exact amount of gas produced to be estimated, so that it can be burned with the exact amount of air to be furnished by the other blast-engine. The ash of the fuel, scorified by the flux, is tapped out every three hours. The outlet-flues of the gas-producer are pierced with holes at their sides, near the inlet, to the steel-refining chamber, and a measured quantity of highly heated air is forced among the gases, insuring their perfect combustion in the flues, which they leave in a downward direction into the heating-chamber, and impinge upon the hearth. The hearth is circular, and is movable laterally and revolving, and is lined with a mixture of calcined dolomite and fluor-spar. The outlet of this chamber conveys the burning gases into the purifying-chamber, lined with sand. In this chamber the pig-metal is first treated with iron-ore, to remove half of its carbon and all of its silicon, and is then tapped out and conveyed by an outside runner to the steel-refining chamber, where it is purified of the remaining carbon and its phosphorus, and when finished is tapped into a ladle and poured into ingot-moulds.

"It is preferred to use molten iron from a blast-furnace, poured into the purifying-chamber, where, by the addition of iron-ore, in two hours half of its carbon and its silicon are removed, so that an operation may be readily made every three hours. The refining of the rest of the carbon and the phosphorus takes about the same time, so that 8 charges a day become praticable. The partially purified iron is treated in the refining-chamber with raw dolomite (about 100 pounds) and 50 pounds of fluor-spar to a ton of pig-iron; and such scrap as is produced is charged and melted with it. The flame, combined with the dolomite and fluor-spar, removes the carbon, sulphur, and phosphorus mostly in the form of vapor, which is combustible and is burned in the outlet-neck of the refining-chamber by air introduced through the holes in the roof, which also burns the gases given off from the iron in the other chamber, thereby insuring the requisite air for their combustion and utilization.

"The purification of the iron into steel may be attained by the use of flame alone when combustible gases are supplied in the exact chemical proportions in which they are required for combustion, and brought together in such a manner that the different molecules which have to enter into combination may readily do so. The proper proportions of hydrogen to carbonic oxide are one to three by volume respectively. This fact is utilized in this furnace, as the gases are produced in a measured quantity and burned with a measured amount of highly heated air in such manner as to secure perfect admixture and combustion before they enter the

heating-chamber, which insures the flame being homogeneous. By its impinging upon the incandescent iron in the hearths the vapor of water produced by burning the hydrogen becomes dissociated, so that its gases remove the phosphorus, sulphur and carbon as combustible vapor, while the silicon is oxidized to silica and becomes slag. From 80 to 90 per cent. of the phosphorus is vaporized, and may be recovered by cooling the gas and condensing it in water.

"A charge of pig-iron, scrap and ore, containing 13½ pounds of phosphorus, treated in this way, with the addition of 75 pounds of dolomite and 50 pounds of fluor-spar to the charge, gave steel with 0 07 per cent. of phosphorus; 2 pounds of the phosphorus went into the slag and about 10½ pounds were vaporized.

"The apparatus is also very economical for making steel from iron that does not contain much phosphorus: 2 cwt., 3 qrs. of coal suffices to make a ton of steel; 2 cwt., 2 qrs. of Republic ore adds ½ cwt. of metal to the weight charge, so that a ton of pig makes 20½ cwt. of steel; white pig-iron makes as good steel as gray: and the labor is but about 35 cents per ton of steel in a furnace of 100 tons capacity a day, when molten iron is used from a blast-furnace, the cost of conversion being about $3.00 per ton.

"One part pig-iron and four parts of scrap are converted to soft steel in 2½ hours, so that 8 melts can be made in 24 hours. The same number of charges are made with pig, and ore and 10 per cent of scrap, when the two chambers are used simultaneously.

"Cold-blast white pig-iron is converted to malleable iron in the furnace in 24 hours without melting, by keeping the flame oxidizing and just below the point of fusion of the iron. This is advantageous for annealing purposes, as but 1 per cent. of carbon is removed in 8 hours.

"Pig-iron containing 3½ per cent. of phosphorus produces steel containing 0.05 per cent. of phosphorus, with 80 to 90 per cent. of the balance of the phosphorus reduced to vapor and condensed, and recovered for fertilizing uses for which it will sell for more than the pig-iron.

"The analysis of the coal pig-iron and dolomite employed, and the steel made at Birmingham, are given in Mr. Brainerd's paper."

A WATER-MANOMETER AND ANEMOMETER.

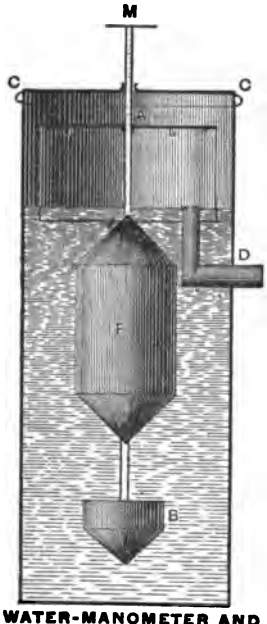
BY J. M. SILLIMAN, LAFAYETTE COLLEGE, EASTON, PA.

(Birmingham Meeting, May, 1888.)

SEVERAL years ago, having occasion to determine the amount of some very slight atmospheric depressions, I devised and had constructed by a skilful tinsmith the manometer shown in the accompanying drawing.

It consists of a cylindrical vessel of tin, 6.6 inches in diameter, through the sides of which, at a distance of about six inches from the top, passes an air-tube D, its upper extremity terminating 3.5 inches from the top. Within the vessel is a combined cylindrical

bell and float. The bell has a diameter of 5.7 inches. The stem is a small brass rod, securely fastened to the top of the bell, through which it passes at A, and is graduated from this point upwards into inches and tenths, the reading of which is taken on the upturned



WATER-MANOMETER AND ANEMOMETER.
Scale ½.

edge of the hole in the cover C C, through which the rod freely passes. This rod continues through the float F, to the bottom of the receptacle for ballast B. For the sides of the drum a thin sheet of tin was selected. The lid C C is perforated with holes for the admission of air above the bell. Hooks are symmetrically placed inside the top of the bell on which pendants may be hung when it is desired to modify the action of the float. The pan at M, for weights, is provided with a socket at its bottom which holds it in place on the end of the rod.

The weight of the bell and attachments = 269.320 grammes, and the weight of the ballast to keep it upright and sink it to the required level = 368.544 grammes. The total floating weight = 637.864 grammes.

In adjusting the manometer for use, it must be placed in a vertical position, and filled with water to within a short distance of the top of

the air-tube D; the bell should then be introduced, and, as its attachments become submerged, the water will rise and flow out of this tube. In this way a uniform water-level is attained for all observations. Sufficient ballast must be placed in the receptacle at the lower extremity to submerge the lower edge of the bell. Small shot are very convenient for this purpose, as a delicate adjustment is necessary. The cover is placed in position, the upper weight-pan slipped on the end of the stem, and the air-tube connected with the space, the depression of which is to be measured.

The principles involved in determining the depression with this instrument are as follows:

r = radius of section of vessel.
r' = radius of section of bell to its inner surface.
r'' = radius of section of stem.
r''' = radius of section of air-tube.
r'''' = radius of section of bell to its outer surface.
a = area of the section of the vessel.
a' = area of the section of the bell to inner surface.
a'' = area of the section of the stem.
a''' = area of the section of the air-tube.
l = mean length of rim of bell.
t = thickness of rim of bell.
lt = area of section of sides of bell.
If pendants are placed inside the bell the area of their section should be included in a'', but if placed outside they should be included in lt.

let h = amount of depression measured in distilled water at 39.1 F.
h' = depression measured by the water in the manometer.
n = distance the bell and stem will be immersed by depression h'.
h'' = distance the water outside the bell sinks under the depression h'.
k = increased height of water from displacement by immersion.
n' = distance the float will sink under depression h'.
w = weight of a unit volume of distilled water at 39.1 F.
w' = weight of a unit volume of the water in the manometer.
n'' = distance which a weight $w'\ h'\ (a'—a'')$, placed on the upper pan of the float, will cause it to sink.
g = number of grammes required on the float to sink it a unit's distance in distilled water at 39.1 F.
g' = number of grammes required to sink it the same distance in the water or other fluid used in the manometer.

The units taken are inches, square inches and grammes; or millimeters, square millimeters and grammes.

We will now have $n\,(lt+a'')\,w' =$ weight of water displaced by bell and stem when immersion increases n units distance; and $w'(a'-a''')h' =$ the force or weight of water causing submergence. Hence,

$$n\,(lt+a'')\,w' = w'\,(a'-a''')\,h', \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

or

$$n = \frac{a'-a'''}{lt+a''}\,h'. \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

Independent of the general rise of water in the vessel from increased immersion of bell and stem due to the depression of the air, we would have under a given depression h', the distance the surface of the water outside the bell sinks below the original level, is to the distance it rises inside the bell above the original level, as the area of the base of the column of water inside the bell is to the area of water-column outside the bell. Hence,

$$h'' : h' - h'' :: a' - a'' - a''' : a - lt - a', \quad . \quad . \quad . \quad . \quad (3)$$

or

$$h'' = \frac{a'-a''-a'''}{a-lt-a''-a'''}\,h'. \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

The general rise of water-level due to immersion is equal to the volume of water displaced divided by its base; but since the rise in the general level of the water adds to the immersion of the rim and stem, the base of the body of water must be taken equal to $a - a'''$. Hence,

$$k = \frac{lt+a''}{a-a'''}\,n = \frac{a'-a''}{a-a'''}\,h'. \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

The total distance the float will sink under the depression h' equals the distance the bell is immersed under this depression, plus the sinking of the water-level outside the bell, minus the rise in the general water-level due to displacement. Hence,

$$n' = n + h'' - k = \left\{\frac{a'-a''}{lt+a''} + \frac{a'-a''-a'''}{a-lt-a''-a'''} - \frac{a'-a''}{a-a'''}\right\} h'. \quad (6)$$

Noting then the distance n' which the float sinks under a depression, the equivalent in head of water employed in the manometer will be

$$h' = \frac{n'}{\dfrac{a'-a''}{lt+a''} + \dfrac{a'-a''-a'''}{a-lt-a''-a'''} - \dfrac{a'-a''}{a-a'''}}. \quad . \quad . \quad . \quad (7)$$

If now the interior of the bell is put in connection with the outer air, and a weight equal to $h' w' (a' - a'')$ be placed on the upper pan, it will displace a volume of water equal to $h' (a' - a'')$ in sinking a distance

$$n'' = n - k = \frac{a' - a''}{lt + a''} \times \frac{a - lt - a'' - a'''}{a - a'''} h', \quad . \quad . \quad . \quad (8)$$

whence,

$$h' = \frac{lt + a''}{a' - a''} \times \frac{a - a'''}{a - lt - a'' - a'''} n''. \quad . \quad . \quad . \quad . \quad (9)$$

From (8) we can also obtain the relation between n'' and n, which is independent of h'.

$$n'' = \frac{a - lt - a'' - a'''}{a - a'''} n. \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (10)$$

From equations (7) and (9) by eliminating h' we obtain the ratio between n', the distance through which the float sinks under any depression, and n'', the distance it sinks when a weight equal to the force of this depression is placed on the upper pan, or

$$n'' = \frac{\frac{a' - a''}{lt + a''} \times \frac{a - lt - a'' - a'''}{a - a'''}}{\frac{a' - a''}{lt + a''} + \frac{a' - a'' - a'''}{a - lt - a'' - a'''} - \frac{a' - a''}{a - a'''}} n'. \quad . \quad . \quad (11)$$

By noting the distance n' the float sinks under any depression, then connecting the bell with the outer air and placing weights on the stem to sink it the distance n'', determined by this equation, we have the measure of the total force of depression in grammes, from which we readily obtain the pressure in grammes per unit of surface, or in the head of water of the standard density. This, being independent both of h' and w', gives us the means of obtaining the force of depression directly without determining the specific gravity of the fluid in the manometer.

After the experimental determination of lt by the method soon to be described, this instrument can be used as a hydrometer; since the specific gravity of the water in the manometer would equal the quotient obtained by dividing the weight required on the float to sink it a unit's distance by the weight which would be required to sink it the same distance in water of the standard density. Let g' equal the weight required to sink the stem a unit's distance, then $n'' g'$ would be the expression for the entire weight as an

equivalent for the force of depression. From this we can obtain the expression for head in distilled water at 39.1 F.

$$h = \frac{a' - a''}{g' n'' w} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (12)$$

Or, if we had first obtained h', we would have for corresponding values of h:

$$h = \frac{w'}{w} h' = \frac{g'}{g} h' \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (13)$$

In the manometer described we have, when there are no pendants,

a	$= 34.21$ square inches.	k	$= .7537\ h'$.
a'	$= 25.5$ square inches.	n'	$= 97.484\ h'$.
a''	$= .0154$ square inches.	h'	$= .01025\ n'$.
a'''	$= .4$ square inches.	n''	$= 96.7363\ h'$.
lt	$= .246$ square inches.	h'	$= .01033\ n''$.
n	$= 97.49\ h'$.	n''	$= .9923\ n'$.
h''	$= .7477\ h'$.	g'	$= 4.300$ grammes.

When a thin sheet of metal is used for the sides of the bell a slight error in the estimation of its thickness t will be so greatly multiplied in obtaining lt as to render the instrument unfit for accurate observations. There is also a slight increase in lt, due to overlapping at the line of junction up the side. The exact mean area lt and thickness of the metal can, however, be determined as follows:

Ascertain by a pignometer the specific gravity of the water used in the manometer; adjust the ballast so the float will be held near the zero point on the scale; place weights on the pan, causing it to sink a considerable distance; note this distance and the weight required, and determine the number of grammes required to sink a unit's distance. This, divided by the weight of a unit volume of the water, will equal the area of the cross-section of the stem and sides of the bell. We will thus have

$$t = \frac{g'}{w' l} - \frac{a''}{l} \text{ or, if no stem, } t = \frac{g'}{w' l} \quad . \quad . \quad . \quad . \quad . \quad . \quad (14)$$

If there are any irregularities in the thickness, these will be detected by this method, and corrections can be made for them.

The tension on the surface of the water along its lines of contact with the surfaces of the bell was so great and so irregular as to render accurate determinations impossible. To overcome this, these surfaces were roughened by rubbing them in a longitudinal direc-

tion with fine emery paper. This was found to make the tension far more uniform, and to greatly diminish it. This advantage was still more evident when the bell was wet above the surface-line of contact. The water then spreads over the wet surface and the tension is practically reduced to zero. This condition is required, in order to obtain the delicate results given below.

In order to test these resistances, and to ascertain with what degree of accuracy the float would adjust itself under any given depression or weight, one of the arms of a delicate balance was extended to five times its length by a very light strip of wood, the outer extremity of which passed over a finely divided scale. The instrument was placed under one of the pans of the scale, and a light weight, partly supported by a fine wire from the arm, rested on the upper pan of the manometer, and, being partly supported by the float, moved up and down with it, communicating its motion to the beam of the scales. The movement was read off on the divided scale at the end of the extended beam where the motion was amplified five-fold.

The surface-tension after roughening and wetting is so slight that 20 milligrammes will promptly move the float, overcoming the slight tension of a surface-line of 36 inches, and the inertia of the float, weighing 637.864 milligrammes, and record its motion at the end of the extended scale-beam. If 10 milligrammes be added to a weight of 20 milligrammes or more, it will manifest itself by additional space.

The responses of the float to the weights of 20 milligrammes and 10 milligrammes prove that under a delicate test it will adjust itself to within less than $\frac{1}{2\,1\,8}$th of an inch from a state of rest, and to within $\frac{1}{480}$th of an inch while in motion; or, in the first instance, to within $\frac{1}{20000}$th of an inch of water, and in the second to within $\frac{1}{40000}$th of an inch.

In the ordinary working of the instrument without any amplifying attachment, it is perfectly reliable to within $\frac{1}{20}$th of an inch, indicating about $\frac{1}{2000}$th of an inch of water.

If a manometer of this kind should be made without an immersed stem or inside pendants, the preceding equations would be simplified by the elimination of a''. Equation (7) would then become

$$h' = \frac{n'}{\frac{a'}{l} + \frac{a' - a'''}{a - l - a'''} - \frac{a'}{a - a'''}}; \quad \ldots \quad (15)$$

Making $h' = \frac{n'}{c}$, we would have

$$\frac{a'}{lt} + \frac{a' - a'''}{a - lt - a'''} - \frac{a'}{a - a'''} = c. \quad . \quad . \quad . \quad . \quad . \quad . \quad (16)$$

By assuming values for a, a' and a''' and solving for lt, the area of the base of the rim of the bell, we will have

$$lt = \frac{a\,a''' - a'''^2 + a\,a' - a'\,a''' + c\,a^2 - 2\,c\,a\,a''' - c\,a'''^2}{2(a' + c\,a - c\,a''')} \pm$$

$$\sqrt{\frac{2aa'a''' - a^2a' + a'a'''^2}{a' + ca - ca'''} + \left(\frac{aa''' - a'''^2 + aa' - a'a''' + ca^2 - 2ca'''a + ca'''^2}{2(a' + ca - ca''')}\right)^2}. \quad (17)$$

Determining this area, we would have the following expression for the radius to the outer surface of the bell:

$$r'''' = \sqrt{\frac{lt}{\pi} + r'^2} \quad ; \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (18)$$

also

$$l = \pi(r'''' + r'), \text{ and } t = r'''' - r'. \quad . \quad . \quad . \quad . \quad . \quad . \quad (19)$$

The above determination of lt is long and complicated.

It is generally practicable, however, to so proportion the manometer as to cause the rise in the general water-level, by immersion, to exactly counteract the sinking of the surface outside the bell from the depression of the air; that is to make

$$h'' = h, \text{ or } \frac{a' - a'''}{a - lt - a'''} = \frac{a'}{a - a'''}. \quad . \quad . \quad . \quad . \quad . \quad . \quad (20)$$

If in this equation we should assume values for a, a' and lt, and make

$$a''' = \frac{a}{2} \pm \sqrt{-a'\,lt + \frac{a^2}{4}} \quad , \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (21)$$

or, assuming values for a', a''' and lt, and place

$$a = a''' + \frac{a'\,lt}{a'''} \quad , \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (22)$$

we would in either case have the proportion

$$n' : h' :: a' : lt \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (23)$$

With this condition we would also have $n = n'$. Assuming either a' or lt and the ratio between n' and h', we would have

$$lt = \frac{h'}{n'}\,a', \text{ or } a' = \frac{n'}{h'}\,lt. \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (24)$$

In designing a manometer for any desired range and accuracy, it would generally be preferable to assume values for a, a' and lt, and determine a''' from equation (21). The negative value of the radical is the only one generally available when the thickness of the sides of the bell is slight. When it is impracticable to make $h'' = k$, equations (24) will still ordinarily give a close approximation for lt or a' for any desired ratio between n' and h'.

Substituting this with the values for the other areas in equation (15), the exact ratio can readily be determined. When it is desired to determine pressure with this instrument, instead of depression, weights can be placed on the upper pan and the pressure determined by formulæ already given.

One defect of the instrument in the form described is its very limited range. If the float should be made in the form of a ring, with ballast at its lower circumference, and the air-tube passing down through the center, its range would be increased.

The range of this instrument, if properly proportioned, can be greatly increased by use of weights on the upper pan. Diminish the ballast and place weights on the upper pan to sink the float to the required initial position on the scale. Then, making connection with the space, the depression of which is required, remove what weights may be necessary to the free movement of the float. The weight removed from the pan divided by that required to sink the float a unit's distance will give the distance through which this weight would sink the float. Adding to this distance the space below the initial point passed over by the float under the depression, and we will have the total space due to this depression.

w'' = number of grammes required on the float to sink it to the initial point.

w''' = weight remaining on the float to permit its free movement under the depression h'.

g' = weight required to sink the float a unit's distance.

n''' = the distance the depression sinks the float below the initial point.

We would then have

$$h' = \frac{\frac{w'' - w'''}{g'} + n'''}{\frac{a'}{lt} + \frac{a' - a'''}{a - lt - a'''} - \frac{a'}{a - a'''}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (25)$$

If the weights employed for this purpose were each equal to that required to sink the bell through the extent of its possible movement, they could be quickly adjusted to measure any depression.

A manometer might be constructed, identical in the principles of its action with that already described, by suspending the bell by a thread passing over a pulley and supporting a counterbalancing weight. The friction of the pulley can be greatly reduced by adding a float ring to the lower end of the bell, of such buoyancy that the weight to be supported by the thread would be only that required to keep the bell in a vertical position. The maximum stability would be obtained by making the float surround the exterior of the bell. A graduated circle of a radius four or five times that of the pulley might be secured to the axis of the pulley to amplify the motion of the bell. The range of this form of instrument can be increased by use of weights just described for the floating bell; but in this case these weights would be applied to directly counterbalance the force of depression.

The area *lt* for any desired result may be comprised in the section of the sides of the bell, or it may be composed of this area and that of the cross-section of outside pendants. Each of these has its advantages and disadvantages as compared with the other.

The first might require a specially prepared plate for the bell; but increasing its thickness would add to its surface-line of contact very much less than where pendants are used. On the other hand, any desired area for *lt* can be readily obtained with pendants, and these can be removed when very delicate tests are required for slight depressions.

This instrument, when connected with the Pitot tube, can also be used as an anemometer, not only as a fixture, as in the Hagemann instrument, but, with the floating bell, as a portable one; and probably more accurate and delicate determinations could be made with it than with the vane anemometer, though the latter is the more convenient instrument for use in the mines.

From the principles given, it will be seen that manometers of this form may be readily constructed, in which the movement of the bell may have any desired relation, from one-thousandth to one thousand times the head of water. If mercury is used, a very sensitive barometer can be constructed on similar principles.

In designing a manometer we should have to consider:

First. The extent of the movement of the bell, which would determine the height of the apparatus.

Second. The force with which it shall adjust itself to within, say, one-thousandth of an inch of water; this force to be slightly in excess of the resistance from tension at surface-line of contact with the water and from the amplifying attachment. This would determine the area a' of the top of the bell.

Third. The range in head of water to be determined by the instrument, which would fix the area lt of the rim of the bell.

The instrument as illustrated should be modified by changing the form of the float and position of the air-tube, as already indicated; by providing overflow tubes connected with the exterior of the bell, or arranging for the overflow over the top; and by adding screw-legs for convenient adjustment to a vertical position.

In Hagemann's anemometer, described in the "Journal of the Franklin Institute," vol. cxxiv, page 185, the bell is suspended by a wire attached to a spiral spring, and equal spaces passed over by a pointer fixed to this wire are taken to indicate equal increase or diminution of depression. There are two errors involved in this estimation of the depression.

First. The spaces passed over by the lower extremity of a suspended spiral spring are not in exact proportion to the weights upon it.

Second. The force of depression acting upon the spring is not in proportion to the depression itself, nor to the actual elongation of the spring under weights corresponding with this depression. There would be a constant change in the ratio between the sinking of the water-level outside the bell and the rise of level within the bell.

In the instrument under consideration, so far as its parts can be determined from the drawing, and with an assumed thickness of .33 millimeters for the sides of the bell, in sinking 10 millimeters it would at its lower limit displace about .9 grammes more water than when in its upper position. In other words, it would require about .9 grammes additional weight on the spring to carry the pointer through the 10 spaces indicating the 10 millimeters depression. With such an instrument the error would vary with the relation between the parts, and would increase with the space passed over.

Both this instrument with the spring, and the manometer after the pattern of the letter-balance recommended by Mr. Hagemann, require for accurate reading the difficult construction of scales of unequal parts. In the latter instrument the lever-arm of the force is constant, while that of the resistance varies as the sine of the angle between a vertical line and the arm carrying the weight.

NOTE ON ARSENIC DETERMINATION.

BY R. C. CANBY, LEADVILLE, COLO.

(Birmingham Meeting, May, 1888.)

THE difficulty of exact neutralization by ammonia of the acid solution obtained in the determination of arsenic by the method of fusion with carbonate of soda and nitrate of potassium, led me to try the effect of neutralization by oxide of zinc. I obtained most satisfactory results, and found a great saving of time. The oxide of zinc being added in excess, there is no delicate testing with litmus paper and tedious alternate adding of dilute ammonia and acid until the proper point is reached. The excess of oxide of zinc, too, seems to hold, from the filter, the gelatinous silica and alumina, which are usually precipitated upon neutralization, and thus to render the washing of the precipitated arseniate of silver almost invariably much more rapid.

Except the substitution of the neutralization by oxide of zinc for the more tedious method by ammonia, the details of the process are practically the same as those published by Mr. Richard Pearce, of Argo, Colorado (*Proceedings of the Colorado Scientific Society*, vol. i.), who shows what very accurate results can be obtained by the fusion process. I have made numerous tests of comparison between the two methods of neutralization, and find that those by the oxide of zinc agree almost exactly with the old method.

Increasing the quantity of nitric acid, even to considerable excess, to be neutralized by the oxide of zinc did not affect the precipitation of arseniate of silver.

The process may be outlined as follows:

Take ½ to 1 gramme finely pulverized ore, and mix in a porcelain crucible with eight to ten times its weight of a mixture of equal parts of carbonate of soda and nitrate of potassium (Kerl. *Metallurgische Probirkunst* gives 4 to 5 nitre to 1½ soda). Gradually bring to fusion and keep so, three to five minutes. Cool. Disintegrate in hot water. Filter and wash with hot water. Acidify the filtrate with nitric acid, and boil to expel carbonic acid and nitrous fumes. Cool. Add an excess of an emulsion of oxide of zinc. The bottom of the beaker should have quite a layer of oxide of zinc, after stirring.

Should an unusually heavy precipitate of gelatinous silica or alumina be formed upon adding the oxide of zinc, it is well to filter before precipitating the arseniate of silver; in which case add additional oxide of zinc to the filtrate before adding the nitrate of silver, especially if the percentage of arsenic is liable to be great.

Add nitrate of silver in slight excess and stir vigorously. Filter and wash with cold water until the filtrate (wash-water) shows no silver. Dissolve, on the filter, the arseniate of silver together with the oxide of zinc in hot dilute nitric acid, allowing the filtrate to pass into a beaker free from chlorine. Cool. Add 1 c.c. ferric sulphate. Titrate with a standardized solution of ammonium sulphocyanide to a pale amber yellow, the same shade of color obtained in standardizing. 108 Ag = 25 As.

CALCULATIONS OF THE AVAILABLE HEAT AND THE REQUIRED DIMENSIONS OF CHIMNEYS, COMBUSTION CHAMBERS, AND GAS BURNERS IN THE USE OF BLAST-FURNACE GASES FOR FIRING BOILERS.

BY FRANK C. ROBERTS, C.E., PHILADELPHIA, PA.

NEGLECTING the hydrogen and hydrocarbons, I will assume the following analysis as a fair average composition, by weight, of the waste gases escaping from a coke-burning blast-furnace:

CO_2	17.25
CO	25.25
N	57.50
	100.00

The 25.25 CO requires for combustion 62.6 of air, or 14.398 of oxygen, necessitating the passage of 48.202 nitrogen into the products of combustion. The latter, therefore, have the following composition:

CO_2	56.9
N	105.7

The total specific heats of these quantities of the respective gases are 12.31 and 25.79, giving for the combined product, 38.1 approximately. That is, the product of combustion will absorb 38.1

calories in rising 1° in temperature. The combustion of the 25.25 CO produces

$$25.25 \times 2400 = 60600$$

calories, and consequently the resulting temperature of combustion is

$$\frac{60600}{38.1} = 1590.5° \text{ C.} = 2894.9° \text{ F.}$$

The foregoing deduction depends upon the burning of the CO by the theoretical quantity of air, or 2.48 units per unit of weight. There is, however, always a considerable excess of air, over and above the theoretical quantity, required for the dilution of the gases. This may be taken at 20 per cent. of the theoretical quantity. On this basis we have the following composition of the products of combustion:

CO_2	56.9
N, ...	105.7
Air, ...	12.5
	175.1

or 1.75 units for each unit of gas. The total specific heats for these quantities are, respectively, 12.31, 25.79, and 2.97, giving the combined product, 41.07. The resulting temperature, computed as before, is

$$1475.5° \text{ C.} = 2687.9° \text{ F.}$$

Carrying the calculation still further, if we assume the air and gas to have, respectively, temperatures of 15.5° and 100° Centigrade,* on entering the burners, the resulting heat of combustion is increased as follows:

17.25 CO_2	×	.2164	×	100	=	373	calories.
25.25 CO	×	.2479	×	100	=	620.5	"
57.50 N	×	.2440	×	100	=	1403.	"
85.00 Air	×	.2377	×	15½	=	303.1	"
Total increase due to initial temperatures, .						2699.6	calories.

The *total available heat of combustion* now being

$$\frac{60600 + 2699.6}{41.07} \quad 1541.2° \text{ C.} = 2774.16° \text{ F.}$$

I will now proceed to determine the least area of chimney-stack required for the effectual consumption of blast-furnace waste gases,

* 60° and 212° F.

the least sectional area of the combustion-chamber, and the least sectional areas of the gas- and air-openings in the gas-burners.

Owing to the peculiar conditions of each and every application of blast-furnace gases, due primarily to their varied composition, it is impossible to arrive at results which would be generally applicable. The best that can be done is to indicate the method by which a correct result for an individual application may be obtained, and to determine, upon certain assumed general conditions, the *least* sectional areas which may be economically employed.

Sectional Area of Chimney.

It is well known that the head produced by a chimney draught is equal to the excess of the weight of a column of air outside and equal to the height of the chimney over a similar column of hot gas within the chimney.

If $H =$ the height of the chimney, $h =$ the head produced in feet of hot gas, $t =$ the absolute temperature of the gas within the chimney, and $t_2 =$ the absolute temperature of the external air, we have

$$h = H\left(0.96\ \frac{t_2}{t} - 1\right)^* \quad \ldots\ldots\ldots\ldots\ldots\ldots (1)$$

As a fair average, and on the basis of heat units, it requires 2.412 pounds of waste gas, of the composition already given, per square foot of boiler heating surface per hour. Taking 11 square feet of heating surface per horse-power, we have 26.53 pounds of gas per horse-power per hour. This gas requires for combustion 22.55 pounds of air, and the products of combustion, as previously deduced, are equal to 46.45 pounds. Hydrogen being present in such small quantities, and the variations in density due to the deviations in pressure of the gases from atmospheric pressure being so slight, we may assume the volume of gases and the products of combustion as being equal, at 32° F., to 12½ cubic feet per pound. Therefore the volume at 32° F. of the products of combustion per horse-power per hour = 580.6 cubic feet, or .1612 cubic feet per second.

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* It will doubtless be noted that I have omitted to take into account, as contributing to the head, the pressure of the gases entering the burners. I have deemed this the better course, however, owing to the continual variations of this factor and the consequent indeterminate nature of the results. The error, at its worst, is on the safe side. Should it be desirable to consider this factor the heads corresponding, in feet of hot gas, to the head in inches of water may be deducted, according to the formula given on page 288 of Rankine's *Steam-Engine*.

Assuming that, in order to obtain the best results,* the temperature of the gases in the chimney uptake is 600°, we have for the volume of the gases in the chimney uptake per horse-power per second

$$V = \text{vol., per sec., at } 32 \quad \frac{1061.2^\circ}{493.2^\circ} = .1612 \times 2.152 = .3469 \text{ cu. ft.}$$

And if S = velocity of current in feet per second in chimney uptake and A = sectional area of chimney in *square feet per horse-power*, then

$$S = \frac{.3469}{A} \quad \ldots\ldots (2)$$

But, according to Peclet,

$$h = \frac{S^2}{2g}\left(1 + \frac{fl}{M}\right) \quad \ldots\ldots (3)$$

where h = head in feet of hot gas; S = velocity in feet per second; l = length of chimney and flue; M = hydraulic mean depth of chimney; f = coefficient of friction for gases moving over sooty surfaces = .012; and g = the acceleration of gravity = 32.17. By transposition

$$S^2 = \frac{2ghM}{M + .012\,l}$$

But also,

$$S^2 = \left(\frac{.3469}{A^2}\right)^2 \quad \therefore \quad \frac{.12034}{A^2} = \frac{64.34hM}{M + .012\,l} \quad \ldots\ldots (4)$$

But $M = \frac{A}{\pi D}$ = area divided by circumference; and for round or square sections $M = \frac{D}{4}$ = one-quarter the diameter.

$$\therefore \quad A = 4\pi M^2$$

Substituting this value in the previous equation, we have

$$1029.44\pi^3 M^5 h - .12034M = .001444l$$

or,

$$3234.088M^5h - .12034M = .001444l \quad \ldots\ldots \ldots\ldots (5)$$

Assuming any desired height of chimney and length of flues and obtaining the corresponding value for the head (h), by equation (1), these values may be substituted in the foregoing equations. A value for M is thus found. But $M = \frac{D}{4}$. $\therefore$ $D = 4M$. Taking the area, corresponding to this diameter, D, we have the

* Rankine.

value of A, = the sectional area of chimney in square feet for *one horse-power* of boiler capacity. Multiplying this value by the total number of horse-power, we have the desired sectional area of the chimney.

Least Sectional Area of the Combustion-Chamber.

By a previous calculation we have found the temperature of combustion equal to 1541.2° C. = 2774.16° F.

The volume per pound at this temperature

$$= 12\tfrac{1}{2} \times \frac{3235.4^\circ}{493.2^\circ} = 81.99 \text{ cubic feet.}$$

The volume per pound at the temperature of 600° in the chimney = 26.9 cubic feet.

The ratio of the former to the latter is

$$\frac{81.99}{26.9} = 3.048$$

But these volumes are directly proportioned to the corresponding areas, and consequently the *least sectional area of the combustion-chamber* is equal to the area of the chimney multiplied by 3.048.

Least Sectional Areas of Gas- and Air-Openings in the Gas-Burners.

We have already found that each unit of gas when consumed gives 1.75 units of products of combustion, and that each unit of gas requires .85 units of air for combustion.

I will first determine the ratio between the volume of one pound of gas at its temperature of admission (212°) and the volume of 1.75 pounds of products of combustion at the temperature in the uptake (600°).

lb. Vol. 1 of gas at 212° $= 12.5 \times \frac{673.2}{493.2} = 17.062$ cubic feet.

Vol. 1.75 lbs. of products at 600° $= 21.875 \times \frac{1061.2}{493.2} = 47.097$ "

And the ratio of the former to the latter is . . $\frac{17.062}{47.097} = 0.362.$

But this ratio is also the ratio between the sectional areas of the gas opening and the uptake, or the *least sectional area of the gas opening*, is equal to the sectional area of the chimney multiplied by 0.362.

Likewise, the ratio between the volume of .85 pound of air, at 60° initial temperature, and the volume of 1.75 pounds of products of combustion, is

$$\frac{11.215}{47.097} = 0.238$$

Consequently, the least sectional area of the air-openings in the gas-burners is equal to that of the chimney multiplied by 0.238.

The contraction of the gas-opening areas by the accumulation of dust contained in the gases must not be overlooked. Due allowance, dependent upon the nature of the gas, should be made for this factor.

As an example of the application of this calculation, let us assume a boiler-stack 110 feet high, with a flue leading to it 10 feet long, *i.e.*, $H = 110$, $l = 120$. Let the temperatures be the same as used in our preceding calculations, then,

$$h = 105.5$$

Inserting the values of h and l in equation (5) we have

$$339740.944M^5 - .12034M = .173280$$

or

$$M^5 - .0000003542\ M = .0000005103, \text{ and } M = 0.05554$$

The diameter $D = 0.22216$ feet and the corresponding area, $A = 0.0387622$ square feet $= 4.582$ square inches *per horse-power of the boiler.*

The least sectional area of the *combustion-chamber* per horse-power =

$$4.582 \times 3.048 = 13.97 \text{ square inches.}$$

The *least sectional area* of the *gas-opening* in the burner *per horse-power* is

$$4.582 \times 0.362 = 1.659 \text{ square inches.}$$

The *least sectional area* of the *air-opening* in the burner *per horse-power* is

$$4.582 \times 0.238 = 1.0905 \text{ square inches.}$$

AUTHOR'S CORRECTION AFTER PRINTING.— For above flue-area (4.582) read 5.582 sq. in. per H. P., making least sectional areas (13.97; 1.659; 1.0905) 17.01; 2.02; and 1.33 sq. in. respectively.

PHOSPHATE SLAG.

BY WILLIAM B. PHILLIPS, PH.D., CHAPEL HILL, N. C.

(Birmingham Meeting, May, 1888.)

IT is proposed in this paper to discuss some of the chemical and physical principles involved in the manufacture and use of this important by-product obtained in the manufacture of steel by the basic process. The name, "phosphate slag," is adopted as a sort of compromise between "Thomas slag" and "tetrabasic phosphate." By the first of these latter names, viz., "Thomas slag," it is known in Germany, Austria, France, and the other continental countries, and in England, and is more easily apprehended by those who have no special knowledge of chemistry, while the name "tetrabasic phosphate," being descriptive of the nature of the slag, is perhaps the best.

It is but seldom that a discovery of such moment to two great industries is to be chronicled in the history of technical science. On the one hand, the manufacture of iron and steel has been greatly benefited, while on the other hand, the effects already noticed, and still to follow, to the agriculturalist, are of scarcely less consequence. The great majority of iron ores in the world, perhaps 90 per cent. of them, are now available for the steel manufacturer, while the manufacturer, and more especially the consumer, of artificial manures is relieved from the fear of the exhaustion of the crude phosphates. We have in phosphate slag, therefore, a product of the highest importance to those who make and use steel and fertilizers. Two great industries, which heretofore have had but little in common, are thus brought into close connection with each other, and the results sure to follow, if one may judge from the history of the past ten years, cannot fail to be of great and increasing value. These results may be compared with those which followed the introduction of the cupriferous Spanish pyrites into England for the manufacture of sulphuric acid. The copper extracted greatly reduced the cost of the acid, hence that of artificial manures. Cheap manures meant better farming and cheaper farm products, and these meant better and cheaper living to nearly all classes of society.

The amount of fertilizers now manufactured in the United States

is about 1,000,000 short tons per annum, valued at about $27,000,000. This will require the consumption of at least 400,000 tons of phosphate rock per annum, valued at $2,000,000. The field for phosphate slag is an inviting one, for it is in the Southern States especially that the greatest consumption and smallest production of fertilizers is to be noticed. It is in the Southern States especially that the new phosphate slag will be produced, for it is here that nature has placed the materials from which the slag is obtained, viz., phosphoritic iron-ores. A brief glance at the statistics of the manufacture and consumption of fertilizers in the States of West Virginia, Virginia, North and South Carolina, Tennessee, Georgia, Florida, Alabama, Mississippi, and Louisiana, will show that while their consumption is over 550,000 tons of fertilizers, their production is not above 225,000 tons. In other words, for every ton of fertilizers they make, valued at, say $20, they import two tons valued at $40.

The State of North Carolina alone spends over $2,000,000 per annum for fertilizers, or for the phosphoric acid contained in them, at least $300,000. It is believed that the ten Southern States just named do not spend less than $3,000,000 yearly for phosphoric acid. Probably eight-tenths of this amount is paid for phosphoric acid from mineral phosphates, leaving two-tenths for the acid from bone, fish-scrap, tankage, etc.

If a new source of phosphoric acid can be had, as good as mineral phosphates, as cheap, and as constant, we may well congratulate the whole country. What, then, can be said, if the new source is better, cheaper, and more constant than mineral phosphates? Yet such is the case. The phosphoric acid in phosphate slag is better than that in mineral phosphates, because it is more readily soluble in the soil; it is cheaper, because for the same amount of money more available plant-food can be purchased; it is more constant, because its original source is more widely diffused, and can be drawn upon for a longer period. This last consideration, however, is of less consequence than the first two. Without robbing the future, or interfering with any of its rights, vested or otherwise, the wisest course is to manage our own affairs and leave posterity to manage its. *Sudor ne ultra crepidam*, which, translated into modern English, may mean, "paddle your own canoe." If, without entailing poverty or distress upon our great-grandchildren, we can secure for ourselves a reduction of 10 to 25 per cent. in the cost of an article we consume, we need not harass ourselves over what they will do.

Let us now inquire what this new source of phosphoric acid is,

how it affects the steel and fertilizer trades, and its value as an article of commerce.

Phosphate slag, or tetrabasic phosphate, is the slag obtained by blowing a blast of air into a mixture of pig-iron and calcined limestone or dolomite contained in a vessel lined with basic material, chiefly calcined limestone, or dolomite. By this means the phosphorus in the pig-iron, whether as phosphide or phosphate, is passed into and absorbed by the lime, forming with it compounds of varying richness in phosphoric acid, but chiefly a compound consisting of 4 parts of lime and 1 part of phosphoric acid. The average analysis of this slag is as follows: German slag: Fleischer.

	Per cent.
Phosphoric acid,	17.5
Lime,	49.6
Magnesia,	4.7
Protoxide of iron,	9.3
Peroxide of iron,	4.1
Alumina,	2.0
Protoxide of manganese,	4.0
Sulphur,	0.5
Sulphuric acid,	0.2
Silica,	7.5

This represents the average composition of the slag from German works, and is given by Fleischer, *Entphosphorung des Eisens durch den Thomas-Prozess.* Berlin, 1886, p. 7.

The average composition of the English slag, as given by Stead and Ridsdale, *Trans. London Chem. Soc.*, 1887, p. 601, is as follows:

	Per cent.
Phosphoric acid,	18.10
Lime,	45.04
Magnesia,	6.42
Protoxide of iron,	2.10
Peroxide of iron,	15.42
Alumina,	1.50
Protoxide of manganese,	3.50
Sulphur,	0.32
Sulphuric acid,	trace
Silica,	5.80
Protoxide of vanadium,	1.35
	99.95

This slag was made by the Northeastern Steel Co., Middlesboro, England, and from it were obtained the crystals to be mentioned later.

G. Moreau (in *La Génie Civil*, August, 1881, quoted in *Eng. and Min. Jour.*, vol. xxxii., No. 8) says that the slag from Teplitz and Witkowitz, Bohemia, had the following composition:

	Per cent.
Phosphoric acid,	27.35
Lime and magnesia,	61.02
Peroxide of iron,	8.19
Protoxide of iron,	1.23
Silica,	2.49

It was produced from a white iron composed as follows:

	Per cent.
Iron,	92.31
Phosphorus,	2.10
Manganese,	2.27
Carbon,	3.12
Silicon,	0.30
Sulphur,	trace

From the analyses quoted it will be seen that the average content of phosphoric acid in phosphate slag is not above 20 per cent. nor below 17 per cent. In Charleston rock (S. C. phosphate) it ranges from 25 per cent. to 28 per cent.; in ordinary Navassa rock, from 27 per cent. to 30 per cent.; in red Navassa, 25 to 27 per cent.; in Canadian apatite, 40 to 43 per cent.; in bones, 22 to 25 per cent.; and in North Carolina phosphate rock, 18 to 20 per cent.

So far as its content in phosphoric acid is concerned, phosphate slag is several per cent. behind the standard crude phosphates, and can, therefore, come into competition with them only if it is better or proportionally cheaper.

Is it better and cheaper? Experiments conducted for the last ten years in Germany, England, and France, and from the last two to three years in the United States, have shown that it far excels as a manure any other crude mineral phosphate. In the condition of an extremely fine powder, at least 75 per cent. of which should pass a sieve of 10,000 meshes per square inch, its phosphoric acid is quickly acted upon by the decomposing agents within the soil, and becomes easly available to plants. It is not within the scope of this paper to discuss any especial points in agricultural chemistry, but it appears necessary to allude, as briefly as possible, to one of the main principles used in the valuation of manures, viz., their content in what is termed "available phosphoric acid."

By this is meant the sum of the phosphoric acid soluble in water, and of the phosphoric acid soluble in a neutral, or slightly acid, or slightly alkaline solution of ammonium citrate. This "available" phosphoric acid is supposed to be of the same, or very nearly the same, value to crops as the soluble phosphoric acid. In the commercial valuation of phosphatic manures a valuation of so many cents per pound is given to the available phosphoric acid. In the southern States 6 cents per pound is allowed. On this basis let us calculate the commercial value of phosphate slag, and compare it with other crude mineral phosphates, and with the acidulated phosphates offered for sale in this country.

In the Report of the Conn. Agric. Expt. Station for 1887, we find that a sample of the slag received from Paul Weidinger & Co., 76 Pine Street, N. Y., contained phosphoric acid 19.48 per cent., of which 6.29 per cent. was "available," not soluble in water but soluble in ammonium citrate. At 6 cents per pound, the value of this slag would be $7.54 per ton. Allowing 2 cents per pound for the balance, *i.e.*, 13.19 per cent., we have as total value $12.81. It is sold f. o. b. in New York for $12.50, and is said to have produced very nearly the same effect on the Indian-corn crop as was produced by a quantity of superphosphate equivalent to it in cost.

Another sample of this slag (Report Conn. Agric. Expt. Station, 1886) yielded 19.57 per cent. available (citrate soluble) phosphoric acid, and would hence have a value per ton of $23.48.

A sample of a cargo of German phosphate slag, imported by Heide & Co., Wilmington, N. C., in 1887 yielded F. B. Dancy (Lab. N. C. Expt. Station) phosphoric acid 17.56 per cent., of which 4.57 per cent. was available (citrate soluble). The value of this calculated as before would be for the available phosphoric acid $5.48, and for the total value per ton $10.28.

It is but seldom more than 2.50 per cent. available or citrate soluble phosphoric acid is contained in finely ground crude mineral phosphates. Taking this as an average, and allowing that 24 per cent. remains undissolved, we have as an average value per ton for the available phosphoric acid $3.00, and as total value $12.60. The f. o. b. cost of such a phosphate, S. C. Floats, for instance, is about $12.50.

The value of phosphate slag, on a basis of 6 cents per pound for the available, and 2 cents for the insoluble phosphoric acid, is thus seen to be quite as high as that of the standard mineral phosphates, and in one case much higher.

As regards acid phosphates with 14 per cent. available, and 3 per cent. insoluble phosphoric acid, the difference may amount to $4 per ton in favor of the acidulated article. It is hardly fair to compare a non-acidulated with an acidulated article, but even in this case by numerous experiments phosphate slag has shown itself of almost equal value with a superphosphate, when the relative cost of the two is considered. Compared with other crude mineral phosphates it has a decided advantage. A ton of phosphate slag costing $12.50, and containing 19.48 per cent. phosphoric acid, would yield to plants 125.8 pounds available phosphoric acid, while to supply the same amount from other phosphates, allowing them even 5 per cent. "available," would cost $15.87, and at 3 per cent. "available" more than $25.

There is a great gain not only in the purchasing power of money as applied to the original phosphate, but also in freight. A ton of phosphate slag will add to the soil as much available phosphoric acid as 2 tons of ordinary phosphate rock, so that a great saving in freight becomes possible, as well as the saving in the price of the original article.

The condition in which the phosphoric acid exists in phosphate slag is a subject upon which I do not here propose to enter. The weight of testimony appears to me to lean decidedly towards the assertion that it exists as tetra-calcium phosphate, $4CaO.P_2O_5$. The tetra-calcium phosphate crystals obtained from the Middlesboro slag by Stead & Ridsdale have less to do in settling this question than the behavior of the slag towards certain reagents, especially towards carbonic acid water, acetic acid and ammonium citrate.

G. Hilgenstock first drew attention to this subject in 1883, and in a paper read before the Union of German Iron Smelters on the 27th June, 1886, said that the ordinary phosphate slag must be a fused mixture of calcium phosphate, calcium silicate and free metallic oxides, and that the whole of the phosphoric acid was present as tetra-calcium phosphate. His opinion was supported by numerous analyses. Others who have written on this subject are Dr. Wedding of Berlin, Wagner, Munro and Wrightson, Groddeck and Broockman (*Stahl und Eisen*, 1884, No. 3, p. 141), Maltzan, Otto and Blum (*Chemiker Zeitung*, 1885, 1886, and 1887), Wyatt, Wiley, Dewey, Goessmann, and Borronicki (*Eng. and Min. Jour.*, 1887), and Stead and Ridsdale (*Jour. London Chem. Soc.*, 1887, p. 601). The work of these last named gentlemen, "Crystals in basic converter slag," is particularly to be praised. They separated at least six differ-

ent kinds of phosphatic crystals from Middlesboro slag of great interest to steel makers, chemists and mineralogists.

The composition of the transparent, slightly yellowish crystals was almost exactly the same as that of tetra-calcium phosphate, viz.:

	Per cent.
Phosphoric acid,	38.04
Lime,	60.13

In a word, therefore, phosphate slag is a mixture of tetra-calcium phosphate with free lime, and the silicates of lime, magnesia, iron, manganese and aluminum.

How does the manufacture of this slag affect the makers and users of steel and of commercial fertilizers?

A full discussion of this important question would require more knowledge, time and space than is at present at my disposal. It presents itself under such a variety of aspects, according to the point of view from which we regard it, it demands such extensive and varied information, from the mining of iron-ore through the manufacture of pig-iron and steel to the grinding and treatment of commercial fertilizers, that instead of a few pages volumes might be written, and a score of men employed in the collection and digestion of data. It is too much for one man to attempt even were he Callon, Bell and Wagner rolled into one, with Edward Atkinson to do the figuring.

There are, however, two considerations which stand out most prominently, viz., the utilization of phosphoretic iron-ores in the manufacture of steel, and the utilization of the slag in the manufacture of artificial manures. From these two depend numerous others, all of which may be referred back to the one or the other of them.

Let us glance at the first, viz., the utilization of phosphoretic iron-ores in steel-making.

I suppose that fully 80 to 90 per cent. of the iron-ores of the world are excluded from steel-works on account of their containing too much phosphorus. As the phosphorus-limit in ordinary steel for structural purposes is taken at 0.10 per cent., and as the phosphorus in the ore goes into the pig, and as two tons of ore are needed to make one ton of pig-iron, it follows that an ore containing more than 0.05 per cent. phosphorus must be regarded with suspicion, when the question of steel-making arises. An ore containing 0.10 per cent. phosphorus is not suitable for steel, or rather *was* not suitable, for now ores containing 2 per cent. of phosphorus can be used.

The importance of this one fact can hardly be overestimated. In order, however, that we may have some specific statements as references, I have compiled from a paper, entitled "The Iron-Ores and Coals of Alabama, Georgia and Tennessee," read by John B. Porter, at the Bethlehem Meeting, American Institute Mining Engineers, May, 1886, and published in the *Transactions*, vol. xv., the phosphorus-content in the various ores of Alabama.

Thus we have:

	Phosphorus.
In Silurian brown ores, average,	0.3116
In red fossil " "	0.2498
In Subcarboniferous " "	0.3675
In carboniferous " "	0.2580
In metamorphic " "	0.2668

The highest phosphorus-content was in the metamorphic ores of Chilton county, viz., 0.5780 per cent., and in the Silurian brown ores of Talladega county, viz., 0.5700 per cent. Assuming that the phosphorus in the ore will reappear in the pig, an assumption very nearly if not altogether true, and calculating the phosphorus to phosphoric acid, we find that the pig from these various ores will contain as follows:

			Phosphoric acid. Per cent.
Pig	from	Silurian brown ores will contain	0.7135
"	"	red fossil " " "	0.5710
"	"	Subcarboniferous " " "	0.8415
"	"	carboniferous " " "	0.5908
"	"	metamorphic " " "	0.6109

The phosphoritic-metamorphic ores of Chilton county, containing 0.5780 per cent. phosphorus, will yield pig, containing 1.3236 per cent. phosphoric acid, and the Silurian brown ores of Talladega county, a pig of 1.3053 per cent. phosphoric acid.

But it takes, on an average, two tons of iron-ore to make one ton of pig-iron, so that, excluding the phosphorus in fuel and flux, the final account will stand:

			Phosphoric acid. Per cent.
Pig	from	Silurian brown ores will contain	1 4270
"	"	red fossil " " "	1.1420
"	"	Subcarboniferous " " "	1.6830
"	"	carboniferous " " "	1.1816
"	"	metamorphic " " "	1.2218

while the pig from the metamorphic ores of Chilton county, containing 1.3236 per cent. phosphoric acid, will contain 2.6472 per cent. phosphoric acid.

According to M. Walrand, a French engineer, with a very considerable experience in the basic process (see abstract in *Engineering and Mining Journal*, vol. xxxiii., No. 13), the most suitable phosphorus-content in pig destined for the basic process is from 2.5 per cent. to 1.2 per cent., equivalent to 5.62 per cent., and 2.74 per cent. phosphoric acid. So that even the most phosphoritic iron-ores of Alabama, whose analyses are given by Dr. Porter, would seem to be just about the lower limit for economical working.

Mr. Alfred F. Brainerd, analytical chemist, Birmingham, Alabama, informs me, in a private communication, April 30th, 1888, that the average phosphorus-content in Alabama pig, examined by him, is about 0.66 per cent., or, of phosphoric acid, 1.5114 per cent.

Fleischer (*Entphosphorung des Eisens durch den Thomas-Prozess, und ihre Bedeutung für die Landwirthschaft*, Berlin, 1886) has collected statistics from eleven German works, making phosphate slag. His results are given in the following table:

Phosphoric acid in charge. Per cent.	Phosphoric acid in pig. Per cent.	Phosphoric acid in slag. Per cent.
2.9	2.5	14.66
	about 2.0	15–20
	1.75–2.0	16–19
	2.5–3.0	16–20
about 2.5	about 2.5	about 18.0
2.08–2.13	2.75–3.0	11.39–15.87
0.5–2.0	2.50–3.0	18.50
3.0	2.5	16–20
about 2.3	2 8–2.9	17–22
about 1.71	about 1.70	12–18
	2.07–3.0	20.2–24.0

Accepting these figures, I have calculated the percentage of phosphoric acid in the charge, in the pig, and in the slag which may be obtained from every 0.109 per cent. of phosphorus in the charge.

The following table exhibits the results:

Table showing the Percentages of Phosphoric Acid in the Charge, in the Pig, and in the Slag for each 0.109 *per cent. of Phosphorus in the charge.*

Per cent. of Phospho-rus in the charge.	Per cent. of Phos. Acid in the charge.	Per cent. Phos. Acid in the Pig.			Per. cent. Phos Acid in the Slag.		
		Maximum	Minimum.	Average.	Maximum	Minimum.	Average.
0.109	0.25	0.55	0.20	0.37	3.70	1.84	2.77
0.218	0 50	1.10	0 41	0.75	7.50	3.73	5.74
0.327	0.75	1.65	0.62	1.13	11.30	5.62	8.46
0.436	1.00	2 20	0.83	1.51	15.10	7.51	11.30
0.545	1.25	2.75	1.03	1.89	18.90	9.41	14.15
0.654	1.50	3.30	1.24	2.27	22.70	11.30	17.00
0.763	1.75	3.85	1.45	2.65	26.50	13.19	19.84
0.872	2.00	4.40	1.66	3.03	30.30	15.08	22.69
0.981	2.25	4.95	1.86	3.40	34 00	16.93	25.46
1.09	2.50	5.50	2.07	3.78	37.80	18.83	28.31
1.119	2.75	6.05	2.28	4.16	41.60	20.71	31.15
1.308	3.00	6.60	2.49	4.54	45.40	22.60	34.00

It will be seen from this table that, on the average, there should be from 0.654 per cent. to 0.763 per cent. of phosphorus in the charge to yield a converter-slag containing from 17 per cent. to 20 per cent. of phosphoric acid.

There are numerous ores in Virginia, Tennessee and Alabama, which would satisfy these conditions.

G. Moreau (in *La Génie Civil* for August, 1881, quoted in *Engineering and Mining Journal*, vol. xxxii., No. 8) is the authority for the statement that the white pig, treated at Teplitz and Witkowitz, Bohemia, an analysis of which has already been given, contained 2.10 per cent. phosphorus (= 4.80 per cent. phosphoric acid), and yielded a slag containing 27.35 per cent. phosphoric acid, or 1 part phosphoric acid, in the pig, yielded 5.70 parts phosphoric acid in the slag. From the data I have been able to get, I suppose that 1 part of phosphoric acid in the pig will, on the average, yield 7.36 parts phosphoric acid in the slag, and thus an ore, containing 0.5 per cent. phosphorus, should yield a converter-slag of 16.85 per cent. phosphoric acid. At some German works the yield is higher: at the works of Fois de Wendel & Co., Hayingen, Lorraine, the ratio is 1 : 9; at the Rothe Erde Works, in Aachen, it is 1 : 9.3; at the Rhein Steel Works, in Ruhrort, it is 1 : 8.75.

The significance of these figures may be appreciated, when we remember that, from the 0.57 per cent. phosphorus ores of Chilton and Talladega counties, Ala., may be produced a converter-slag, containing from 19.21 to 24.27 per cent. phosphoric acid.

The value of the slag as a by-product stands in intimate con-

nection with its value as a manure, and its value as a manure depends upon whether it can most profitably be converted into a superphosphate or precipitated phosphate, or used as finely ground raw phosphate. This is not the place to speak of its conversion into super or precipitated phosphate, except to mention that such a process is successfully carried out in Germany. Nor shall I speak of the various processes by which a concentration of its phosphoric acid is attained by mechanical means, such, for instance, as by Scheibler's process. It seems to me that its chiefest use and chiefest value lies in the direction of its application to the soil as fine ground crude phosphate. As such, it must come into competition with other fine ground crude phosphates. Compared with them, it has nothing to fear. It enters the struggle with many and great advantages, and it will rest with the skill and perseverance of those who introduce it, and not with the slag itself, if it should not succeed in winning an important position in the estimation of agriculturists.

In concluding, I cannot forbear the expression of my indebtedness to the pages of the *Engineering and Mining Journal*, which, for the last eight or nine years, have contained numerous and valuable references to the various details of the Basic process.

THE GRADING OF BIRMINGHAM PIG-IRON.

BY KENNETH ROBERTSON, BIRMINGHAM, ALA.

(Birmingham Meeting, May, 1888.)

ALL strangers visiting this district are struck with the peculiar manner in which the pig-iron is graded. There are eleven regular grades, besides which, when gray forge is ordered, one-half of Nos. 1 and 2 Mill are shipped. Occasionally there is another grade known as Silvery Mill, which is made so seldom that I cannot describe it, and have no sample to exhibit.

Most of you have found it difficult to grade properly and uniformly under the simpler system which obtains elsewhere, and can consequently readily imagine the increased difficulty with us. Each furnace employs an "expert," and even with this precaution, the system is not conducive, at all times, to amicable relations between buyers and sellers. I am told that it was adopted at the time Southern irons were seeking a market; but it still remains, although the time has come when our iron is sought for and has obtained for itself a place in the markets of the country.

The grades are as follows:

No. 1 *Foundry*, a large-grained, dark-colored iron with crystallization extending well out to the edges of the pig. In my experience but little of it is made, and I am inclined to regard it more as a freak than a product. An average of three analyses shows 3.66 per cent. silicon in this grade.

No. 2 *Foundry* is the equivalent of a No. 1 Foundry at the North. An average of eighteen analyses gives 3.02 per cent. silicon.

No. 2½ *Foundry* corresponds to No. 2 Foundry elsewhere. An average of eight analyses shows 3.02 per cent. silicon.

No. 1 *Mill* is also known as No. 3 Foundry, and in it are included irons which are not quite good enough for 2½ Foundry, and also those which are equal to what is known as gray forge in the Lehigh Valley and vicinity. The best of this iron is used for foundry purposes. An average of four analyses gives 2.87 per cent. silicon.

No. 2 *Mill* is between 1 Mill and Mottled, and contains 2.44 per cent. silicon as an average.

No. 1 *C.* is open-grained silver-gray. I have but one analysis, which shows 5.25 per cent. silicon.

No. 2 *C.* is close-grained silver-gray. Average of three analyses, 7.09 per cent. silicon.

No. 1 *Bright* is a foundry-iron which is light in color but open grained. It is made by every furnace in every district, at times; but it is only in this section that it is separated from the foundry-irons. Elsewhere it would be shipped as No. 1 Foundry. Average of three analyses, 3.69 per cent. silicon.

No. 2 *Bright* is one grade lower; is closer-grained; and the average of fourteen analyses is 3.11 per cent. silicon. Elsewhere it would be a No. 2 Foundry.

To complete the number, we have Mottled and White, which are the same here as elsewhere.

An idea has been prevalent for a long time that Southern irons are highly siliconized and weak; that the product of the furnaces is not foundry, but chiefly of the lower grades; and that the lower grades are sold with difficulty. The preceding analyses show that the foundry-irons do not contain more silicon than irons of the same grade in other districts; the mill-irons are higher in silicon than those of Glendon and Andover, but they are sold without difficulty. As to the product of the furnaces, I will give the percentages of each grade which one of the furnaces under my charge made during ten months' working under very disadvantageous circumstances. Other furnaces in the district have undoubtedly done much better; and these figures are not given as typical, but

merely to show that we do make foundry-iron, and that the greater portion of our product is not of lower grades.

The average of twenty-seven determinations of phosphorus is 0.66 per cent.

Percentages of Grades made.

No. 1 Foundry,	0.27
No. 2 Foundry,	26.23
No. 2½ Foundry,	19.48
No. 1 Mill,	33.82
No. 2 Mill,	6.85
No. 1 C.,	0.56
No. 2 C.,	1.39
No. 1 Bright,	6.07
No. 2 Bright,	2.76
Mottled,	2.38
White,	0.19
	100.00

Calling the Bright irons foundry, which they are, the proportion of foundry-iron made was 54.81 per cent.; probably half the 1 Mill would have been classed as foundry elsewhere. These results are not considered as the *ne-plus-ultra* of furnace-work, but will show what we are doing, and also that the impression that but little foundry-iron is made here was erroneous.

THE FEASIBILITY OF USING CHEAPER FUELS IN THE BLAST-FURNACE.

BY JACOB T. WAINWRIGHT, PITTSBURGH, PA.

(Birmingham Meeting, May, 1888.)

THE object of this paper is to describe some efforts and observations by the writer relating to this subject.

Although the usual type of blast-furnace is a most efficient device for smelting and melting purposes, it is not suitable for utilizing the cheaper grades of fuel, such as coal-slack and certain gaseous fuels.

This problem not only involves the consideration of the furnace as a suitable apparatus, but also embraces the consideration of the chemical and calorific properties of the proposed fuel.

In the hearth of the ordinary blast-furnace an extremely high temperature is maintained. This extreme temperature is due to the complete combustion of the fuel in the hearth, and before the gases reach the zone of fusion their temperature is reduced, by the absorption of carbon, 2300 degrees Fah.

In a smelting-furnace, the maintenance of this extreme temperature at the hearth becomes useless when it can be reduced without

altering the condition, or calorific effect of the reducing gases resulting from combustion. This can be accomplished by providing an auxiliary combustion-chamber, and passing the gases thence into the hearth and upward through the stack.

By this method the temperature of the hearth is under control, and may be elevated when necessary. Also, scaffolding is avoided, since a melting temperature is continuously maintained. Moreover, by this method impurities that may be contained in the fuel are oxidized before being brought into contact with the metal, which seems to prevent their absorption by the latter.

By means of a structural device for supporting the burden, I have operated a furnace of this description. This furnace consisted of a vertical shaft, with an auxiliary combustion-chamber leading into the hearth. Above this hearth, wrought-iron tubes encircled by fire-clay tiles were placed across the shaft to form a support for the furnace-charge, with sufficient space between to allow the gases to ascend and permeate the burden, and also to allow molten material to descend into the hearth. These tubes were cooled by a blast of air passing through them, the object being to prevent their being destroyed by the extreme heat in the interior of the furnace.

In this furnace I used natural gas as fuel, and demonstrated that pig-iron could be melted without charging any fuel with the metal, and that ore could be smelted with about 15 per cent. of the amount of coke usually employed.

The feasibility of separating the greater portion of the fuel from the burden having thus been demonstrated, the question suggests itself, why not use a cheap fuel in such a combustion-chamber, and what cheap fuels are suitable for this purpose?

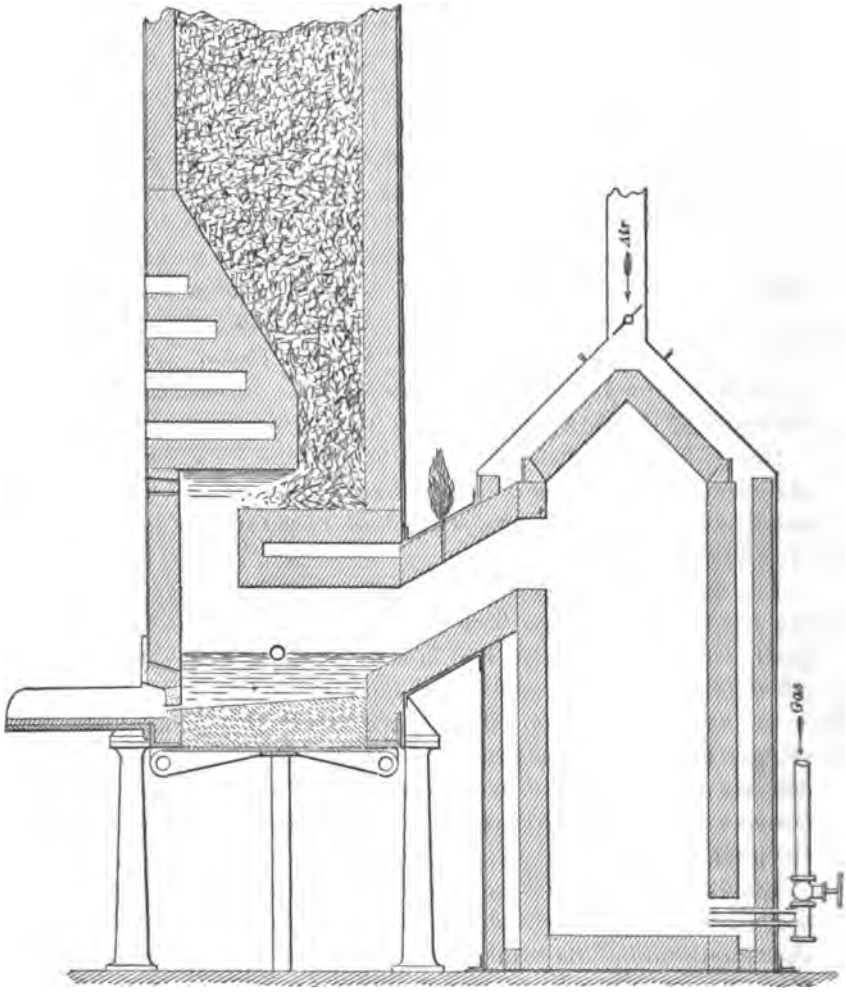
In answer I would say, that a carbonaceous fuel which does not contain too large a proportion of diluents will answer. The usual diluents are combined oxygen, hydrogen, and nitrogen. Hydrogen becomes a diluent in this case, because at high temperatures and in the presence of free carbon it will not combine with oxygen.

In reviewing the various cheap fuels, it may be remarked that coal-slack is a most desirable fuel for this purpose, since in its use the temperature of the reducing gases would not be greatly lowered. Next in order might be mentioned petroleum, which contains about 84 per cent. carbon, 14 per cent. hydrogen, and 2 per cent. oxygen. In this case, the temperature of the reducing gases would be 700 degrees Fah. lower than from coke. Natural gas, which contains about 75 per cent. carbon and 25 per cent. hydrogen, would produce

reducing gases 1100 degrees Fah. lower in temperature than from coke. These deficiencies in temperature, however, may be overcome by means of hot-blast stoves.

Carbonic oxide, producer-gas, water-gas, and hydrogen, are unfit for this purpose, because they do not contain carbon which is not combined with oxygen.

FIG. 1.



Cupola Furnace Using Gaseous Fuel.

The following table, relating to the combustion of various kinds of fuel, shows some interesting facts from which the relative value of these fuels can be readily deduced:

TABLE RELATING TO THE COMBUSTION OF FUEL.

	Combustible.	Weight of 1 cubic foot of combustible.	Chemical symbol.	Elements of combustible by weight.				Lbs. of oxygen for combustion of 1 lb. of combustible.	Lbs. of dry air for combustion of 1 lb. of combustible.	Cubic feet of dry air for combustion of 1 lb. of combustible.	Cubic feet of dry air for combustion of 1 cubic ft. of combustible.	Total heat units (British Association) generated by 1 lb. of combustible.	Elevation of temperature in degrees Fah. for combustion in oxygen.	Elevation of temperature in degrees Fah. for combustion in dry air.	After passing the hot gases through a bed of coke, the (H) is set free and the (CO_2) converted to (CO).			
				Nitrogen.	Carbon.	Oxygen.	Hydrogen.								Lbs. of carbon absorbed from bed of coke per lb. of combustible.	Total heat units (British Association) generated by 1 lb. of combustible and the bed of coke.	Proportion of heat units generated by combustible.	Elevation of temperature in degs. Fah. for combustion in dry air.
Complete combustion of combustible. The (C) to (CO_2) the (H) to (H_2O).	Carbon.		C		1.000			2.666	11.597	143.70		14500	19536°	4966°	1.000	8800	$\frac{5}{10}$	2650°
	Producer gas (from dry air).	.0782		.657	.147	.196		.196	.852	10.56	.826	1483	1359°	1185°	.147	647	$\frac{1}{10}$	1601°
	Carbonous oxide.	.0780	CO		.428	.572		.572	2.488	30.85	2.406	4324	13614°	5510°	.428	1883	$\frac{1}{10}$	1968°
	Water gas.	.0418	H_2+CO		.400	.533	.067	1.068	4.647	57.62	2.408	8190	15682°	5616°	.801	3524	$\frac{1}{10}$	1055°
	Petroleum.	54.81 for liq.			.840	.020	.140	3.327	14.472	179.45	9833.9	20661	16832°	5238°	1.671	10960	$\frac{3}{10}$	2367°
	{ Various liquid } { Hydrocarbons. }										from.... to	19000 } 21700 }						
	Olefiant gas.	.0795	CH_2		.857		.143	3.426	14.903	184.79	14.690	21297	16995°	5257°	1.713	11308	$\frac{3}{10}$	2379°
	Marsh gas (natural gas).	.0611	CH_4		.750		.250	3.995	17.378	215.48	13.166	26383	16132°	5386°	2.247	13187	$\frac{2}{10}$	2263°
	Hydrogen gas.	.0056	H				1.000	7.980	34.713	430.44	2.410	62032	14376°	5727°	5.986	26338	$\frac{0}{10}$	1972°
Incomplete combustion of combustible. The (C) to (CO) the (H) set free.	Carbon.							1.333	5.798	71.85		4400	7697°	2650°	0	4400	$\frac{1}{10}$	2650°
	Producer gas (from dry air).							0	0	0	0	0	0	0	0	0	$\frac{0}{10}$	0
	Carbonous oxide.							0	0	0	0	0	0	0	0	0	$\frac{0}{10}$	0
	Water gas.							0	0	0	0	0	0	0	0	0	$\frac{0}{10}$	0
	Petroleum.							1.093	4.754	58.95	3234.4	3608	3816°	1962°	0	3608	$\frac{1}{10}$	1962°
	{ Various liquid } { Hydrocarbons. }																	
	Olefiant gas.							1.142	4.967	61.59	4.896	3771	3858°	1974°	0	3771	$\frac{1}{10}$	1974°
	Marsh gas (natural gas).							1.000	4.350	53.94	3.296	3300	2576°	1573°	0	3300	$\frac{1}{10}$	1573°
	Hydrogen gas.							0	0	0	0	0	0°	0	0	0	$\frac{0}{10}$	0

DATA USED IN COMPUTING THIS TABLE.

Weight of 1 cubic foot of	O	.0892	pound.
" "	CO	.0780	"
" "	CH_2	.0795	"
" "	CH_4	.0611	"
" "	H	.0056	"
" "	nitrogen	.0784	"
" "	air	.0807	"

Specific heat of water,	1.0000
" hydrogen,	3.4090
" nitrogen,	.2438
" CO_2	.20246
" CO,	.2450
" steam (H_2O),	.4805

Atomic weight of oxygen,	15.96
" hydrogen,	1.00
" nitrogen,	14.01
" carbon,	11.97

Total heat units (British Association) resulting from combustion of:

One pound of C to make CO_2 =	14500
" C " CO, =	4400
" H " H_2O, —	62032

Composition of dry air by weight:

Oxygen,	.23
Hydrogen,	.77
	100

DETERMINATION OF PHOSPHORUS IN IRON AND STEEL.

BY PORTER W. SHIMER, M.E., EASTON, PA.

(Birmingham Meeting, May, 1888.)

THE solution used in the following method for phosphorus-determination is the filtrate obtained in the nitric and sulphuric acid method for the determination of silicon (*Transactions*, vii., 346). This filtrate, being easily and quickly obtained, and always quite free from silica, is a desirable one for phosphorus-precipitation. In order, however, to get all the phosphorus in a precipitable form the solution must be made under conditions more strongly oxidizing than simple solution in boiling nitric acid. In a solution thus

made I have found that the presence of a moderate amount of free sulphuric acid does not prevent complete precipitation of the phosphorus.

The method in detail is as follows: Dissolve 1 grm. of iron in 20 c.c. HNO_3, sp. gr. 1.20. Add to the boiling solution 10 c.c. potassium permanganate solution (20 grams to 1000 c.c. water, filtered), $2\frac{1}{2}$ c.c. at a time. After a few minutes, when the permanganate is decomposed, leaving a heavy precipitate of MnO_2, add 5 c.c. HCl, 1.12 sp. gr. When the effervescence due to the escape of chlorine has ceased, add a mixture of 5 c.c. strong H_2SO_4 and 5 c.c. water. Remove the watch-glass and evaporate till fumes of SO_3 begin to come off. Allow to cool and add 5 c.c. HNO_3, 1.20 sp. gr., and enough water to dissolve the residue. Boil till all iron salts are dissolved and filter, washing with water alone. Set aside the filtrate for phosphorus-determination. It need not be concentrated by evaporation when unnecessary excess of wash-water is avoided. For silicon the residue should be washed a few times with HCl and water to remove possible traces of iron, but these washings should not be added to the phosphorus solution. The latter is heated to 80° C. and 50 c.c. ammonium molybdate solution is added (5 grms. MoO_3, 20 c.c. NH_4HO, sp. gr. 0.96, and 30 c.c. HNO_3, sp. gr. 1.20). Heat to 60° C. till the supernatant liquid is perfectly clear. When the solution is not too dilute, precipitation is complete in less than one hour. The yellow precipitate is filtered off, washed with acid ammonium nitrate solution, dissolved in ammonia and precipitated by magnesia mixture. The ammoniacal solution is more generally colorless and clear than that obtained by the usual method. Whenever this solution is not perfectly clear and colorless, it is my custom to dissolve in HCl whatever gelatinous residue there may be upon the paper, allowing the acid solution to drop into the ammoniacal filtrate below until it becomes acid, thus throwing down the yellow precipitate. Add a slight excess of ammonia and boil for a few minutes. Filter off the precipitate, dissolve it on the paper in a few drops of HNO_3, and add ammonium molybdate solution. Add the ammoniacal solution of the yellow precipitate thus obtained to the main solution. The latter is then precipitated with magnesia mixture in the usual manner.

When solution of the iron is made as above, no phosphorus can escape complete oxidation, for it is subjected to the following oxidizing agencies: Solution in boiling HNO_3, action of potassium permanganate solution, action of chlorine given off by reaction of HCl

on precipitate of MnO_2, action of chlorine given off by mixture of strong HCl and HNO_3, evaporation and heating till fumes of SO_3 appear, solution of residue in water and HNO_3. These agencies may also be desirable for the oxidation of silicon, for there is reason to suspect that this element is not easily oxidized in some irons.

The following are some comparative results obtained. The "standard" method is the usual one of solution in HNO_3; evaporation to hard dryness; solution of residue in HCl; replacement of HCl by HNO_3, precipitation by molybdate solution, and final precipitation by magnesia mixture.

	Standard Method. Phos. per cent.	Method Above Described. Phos. per cent.
A,	1.75	1.74
	.....	1.76
	.....	1.76
B,	3.623	3.639
	3.641	3.637
C,	0.560	0.558
D,	0.394	0.397
E,	0.582	0.588
F,	0.073	0.072
	.....	0.074
G,	0.749	0.749

In the Bessemer iron ("F") 5 grms. iron and 8 c.c. H_2SO_4 were used. Sample "B" is a white iron, containing 2.27 per cent. combined carbon and 0.08 per cent. graphite. In sample "G," Mr. Lee S. Clymer, Chemist of the Durham Iron Works, obtained by the usual acetate method 0.743 per cent., and, by applying the acetate method to the filtrate from the silicon as above, he obtained 0.743, 0.740 per cent. phosphorus.

The exactness with which the kind and amount of free acid can be controlled in this solution for molybdate-precipitation, and the comparative purity of the yellow precipitate in case of titanic irons, would probably make it well adapted either for weighing directly or for solution, reduction, and titration, according to the method described by F. A. Emmerton, *Transactions*, xv., 93.

N. H. Muhlenberg and Thomas M. Drown (*Transactions*, x., 85, 329) evaporate the solution of the iron in nitric and sulphuric acids to dryness and heat till SO_3 fumes cease coming off. The phosphorus-results obtained in this way are accurate, but the silicon-results are too high. It is unnecessary, however, to drive off dense volumes of these irritating fumes, for complete precipitation depends, not upon the absence of free sulphuric acid, but upon

complete oxidation of phosphorus, and of the carbon compounds resulting from solution of combined carbon in nitric acid. Moreover, a certain amount of free sulphuric acid is necessary for a good silicon determination.

It may be interesting to add the results of a few experiments with a molybdate solution containing sulphuric acid in place of nitric acid. The solution for phosphorus-precipitation was made as above, with the exception that the residue on evaporation with H_2SO_4 was dissolved in water alone, instead of water and nitric acid. This solution was heated to 80° C. and precipitated by a molybdate solution, made by dissolving 5 grms. MoO_3 in 20 c.c. NH_4HO, sp. gr. 0.96, and adding 30 c.c. dilute H_2SO_4 (1 part strong H_2SO_4 to 4 parts water). After heating for one-half-hour to 60°C. precipitation was complete. The following are the results:

	By Standard Method. Phos. per cent.	By Use of Sulphuric Molybdate Solution. Phos. per cent.
A,	1.75	1.75
B,	3.63	3.64

It will be noticed that in these precipitations sulphuric acid and ammonium sulphate completely take the place of nitric acid and ammonium nitrate. Another experiment on the iron marked "B," in which, however, 25 c.c. more free sulphuric acid was present than in the above tests, gave only 3.09 per cent. phosphorus, showing that a too large excess of free sulphuric acid, like too large an excess of free nitric acid, prevents complete precipitation.

MINING IN SOFT ORE-BODIES AT LOW MOOR.

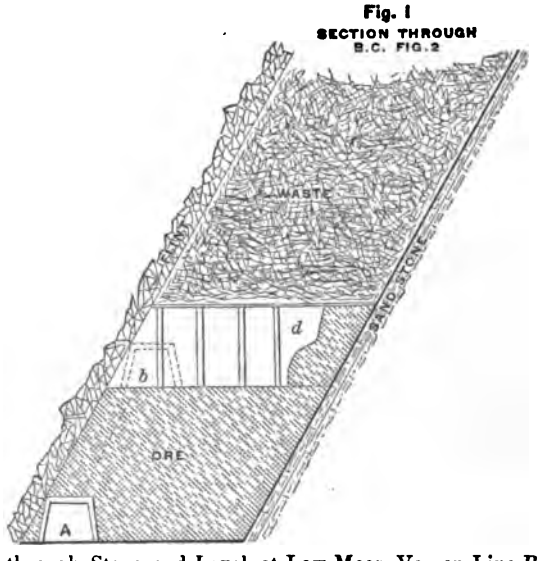
BY W. S. HUNGERFORD, LOW MOOR, VA.

(Birmingham Meeting, May, 1888.)

As several papers on the subject of mining in soft ore-bodies have recently appeared in the *Transactions* and in the *Engineering and Mining Journal*, the following brief account of the method introduced by the writer at Low Moor, Va., may be of interest. It is in some respects similar to the system at Longdale, Va., referred to in Mr. Rothwell's paper, though differing in many of its details. That system has been employed by the Longdale Iron Company for many years, and to that company the writer is indebted for many

valuable suggestions. The method here described will also be recognized as embodying the same general principles as that described by Mr. Goudie in the *Engineering and Mining Journal* and the one proposed in Mr. Rothwell's paper above mentioned. It is not claimed that the system here described contains any improvements over those above referred to; and it is given only as showing actual experience with this method of mining.

The ore-bodies at Low Moor lie in a vein which, for the purposes of this description, is supposed to have a dip of 60° and a thickness of 30 feet. These are in fact somewhere near its average conditions, though much irregularity exists; the dip in different portions of the

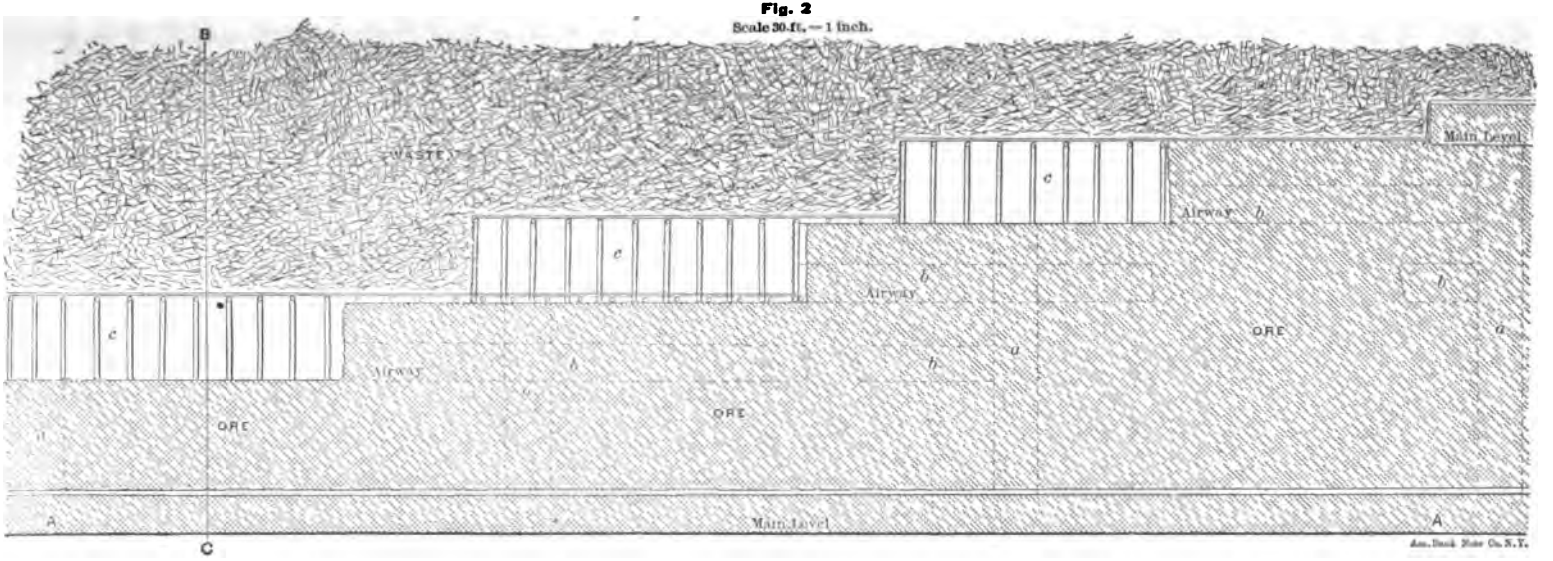


Cross-section through Stope and Level, at Low Moor, Va., on Line B C of Fig. 2.

mines varying from vertical to nearly horizontal, and the thickness from 10 to 40 and even 60 feet. The ore is generally soft, and all drifts require close timbering. If several drifts near each other, either one above the other or on the same level, are left standing long, the ore will "creep," throwing timbers out of line and even heaving the bottom badly. In the portion of the vein here described the hanging-wall is a band of broken flint and clay and the foot-wall is sand-rock, with or without loose sand. The vein here is really inverted, the sand-rock being usually the hanging-wall of the veins of this region.

Main levels (A, Figs. 1 and 2) are driven in the vein along its

Fig. 2
Scale 30 ft. — 1 inch.



Longitudinal Vertical Projection, showing Method of Ore-extraction at Low Moor, Va.

strike, 60 to 80 feet apart vertically, starting from the surface on the side of the hill or from a hoisting-shaft in the valley.

They are usually on the flint wall (see Fig. 1) whether it be the foot or the hanging, because the flint is a better guide in following the vein. These levels are driven along the vein to such a distance from the hoisting-shaft, as may be required to reach all the ore which it is intended to raise through that shaft—in some instances over half a mile. Hoisting-shafts might have been sunk at more frequent intervals, but the mountainous character of the country and the difficulty of surface-transportation seemed to determine their location. While the main-levels are being driven the pillar between two levels is usually left untouched, except by up-raises connecting the two levels every 400 to 600 feet for the purpose of ventilation. When the levels are completed, the portions of two levels farthest from the hoisting-shafts are connected with up-raises (*a*, *a*, Fig. 2), 60 to 75 feet apart, two or three up-raises are joined by air-drifts (*b*, *b*, Figs. 1 and 2) and the ground is ready for stoping. The timber in the air-drifts is recovered in stoping. The stopes are 12 to 15 feet high, each pillar between two main levels making 4 to 6 stopes. As soon as a stope is worked out for 40 to 60 feet along the vein a floor is laid, consisting of sills covered with refuse timber or slabs, and the props are shot down. The waste material from above packs solidly upon this floor, and in a short time the next lower stope can be worked, using the floor previously laid as a roof to hold the waste material from the ore. A stope 40 to 60 feet long, measured along the vein, is begun in the drifts *b*, *b*, by first mining the ore above the drift-timbers till the floor of the next stope above is reached, and setting props. The face of the ore for the length of the stope is then mined back to the opposite wall, as shown at *d*, Fig. 1. The ore is dumped into the chutes, *a*, *a*, *a*, and drawn from them into cars on the main levels A, and hauled by mules to the surface or to the hoisting-shaft.

Fig. 2 represents a vertical projection along the strike of the vein, showing three stopes. The middle one is completed and the floor is laid ready for shooting down the timbers.

Fig. 1 represents a section on line B C of Fig. 2.

When the vein is 12 feet or less in thickness, so that a single prop will reach from wall to wall, the method is somewhat modified. A stope-drift is driven a short distance above the main level, parallel with it, and connected with it by chutes at intervals of about 50 feet. The ore is then stoped from this drift to the next upper

main level, props being placed from wall to wall. These props are eventually shot down, and the waste material is caught by a horizontal floor which will serve as a roof for the next lower stope, as before. In this way the timber for a number of floors is saved, but the modification is only of advantage where the vein is narrow enough to permit a single prop to reach from wall to wall, and where, moreover, the hanging-wall is fairly good.

In this way all the ore is mined; no filling is required and the work is comparatively safe.

During the past four years, with an average of about 200 men employed, there have been but two fatal accidents. In one of these cases a boy fell down a shaft; in the other a man was crushed by a fall of ore. The latter accident happened before the present system was adopted.

The country-rock is soft and readily fills in the cavities formed. It breaks to the surface, which, in places, is 500 feet above the work. The surface-damage is here no consideration, nor does the surface-water give much trouble, though in some instances arrangements must be made to conduct it around the surface-breaks.

This method has been in use at Low Moor for about three years. The annual production has been about 50,000 tons of clean ore; or, allowing for loss in washing, more than double that amount of material, all of it from underground mining.

THE PETITE ANSE SALT-MINE.

BY RICHARD A. POMEROY, NEW IBERIA, LA.

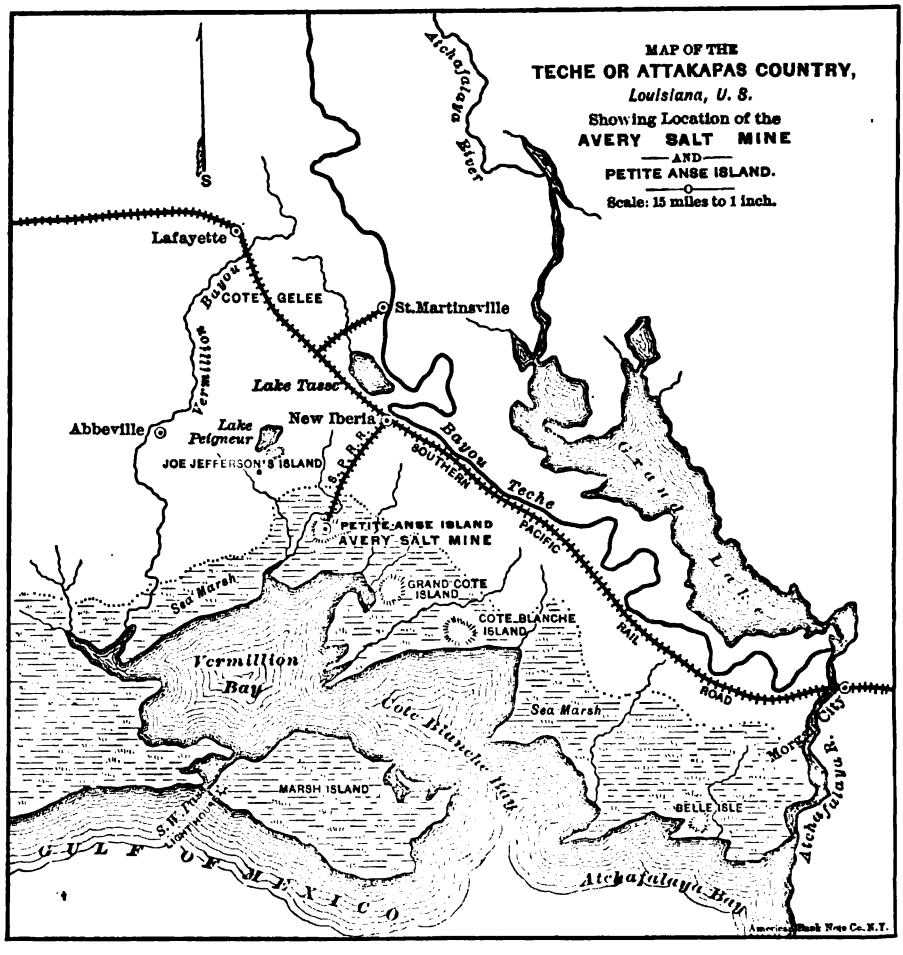
(Birmingham Meeting, May, 1888.)

THIS mine, known also as the Avery Mine, is situated on Petite Anse Island, Iberia Parish, Louisiana. The island is about four miles from Vermillion Bay, an arm of the Gulf of Mexico, and is one of four islands or hills surrounded by the sea-marsh, as shown in the accompanying map. (Fig. 1.) A fifth, Joe Jefferson's Island, or hill, is some distance from the sea-marsh. These five islands: Belle Isle, Cote Blanche, Grand Cote, Petite Anse and Jefferson, appear to be the tops of submerged hills, and occupy a line having nearly a northwest trend. To the north are a series of hillocks that extend at intervals to the hills of Northern Louisiana, which in turn appear to run into the Ozarks of Arkansas. Near the mine there is sandstone, also a thick vein of lignite. Bones of the mastodon have been

found in considerable quantity in this neighborhood, and some specimens have been sent to the Smithsonian Institution.

I will not give here a geological description of the country, since this has been abundantly done already by Hilgard, Goessmann, Thomassy, and other able writers. It will, doubtless, be sufficient to

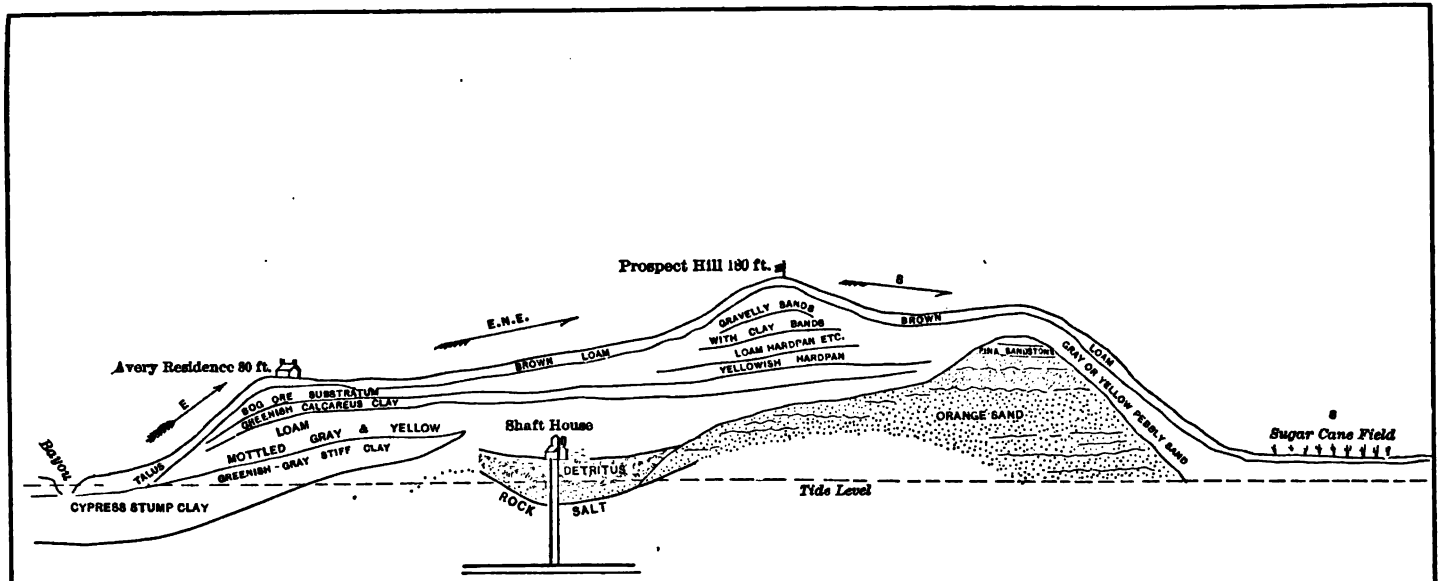
FIG. 1.



call the reader's attention to the accompanying geological section of Petite Anse Island (Fig. 2), copied from Prof. E. W. Hilgard's map.

The salt-mine and Petite Anse Island are owned by the Avery family, an old and honored family in Louisiana. The New Iberia Salt Company is the lessee, J. C. Haskell being general superintendent, S. L. McCalla, mining engineer, and John Hamilton, mill

FIG. 2.



GEOLOGICAL SECTION OF PETITE ANSE ISLAND, LOUISIANA.

(Copied from Prof. E. W. Hilgard's Map.)

American Bank Note Co. N.Y.

superintendent. As far back as the beginning of the present century salt-springs were known to exist on the island, wells had been dug and salt made for local consumption. During the war much of the southern country was cut off from its supply of salt, and the island became a busy place for its manufacture. The supply of brine proving insufficient, an old well was cleaned out under the direction of Mr. John M. Avery. In digging this well still deeper the negro at work told his master that he had struck a log and could go no further, whereupon Mr. Avery descended and discovered a hard rocky substance, which upon examination proved to be pure rock-salt. This, the first discovery of the salt, was made at a depth of only 16 feet. It put an end to the manufacture by evaporation. Pits of rectangular form were sunk and the earth was stripped from the salt, which was blasted out by underhand stoping. Wagons by the thousand were used to transport the rock-salt. Upon the occupation of the country by the Federal forces the work of mining was suspended.

The depth of the salt has never been determined, but the surface-area underlain by the deposit is estimated to be about 150 acres. The salt is very pure and compares favorably with various rock-salts and with products of evaporation-pans, springs, wells and lakes. The following table of analyses of salt speaks for itself. In the analysis of the Avery salt, many eminent chemists agree.

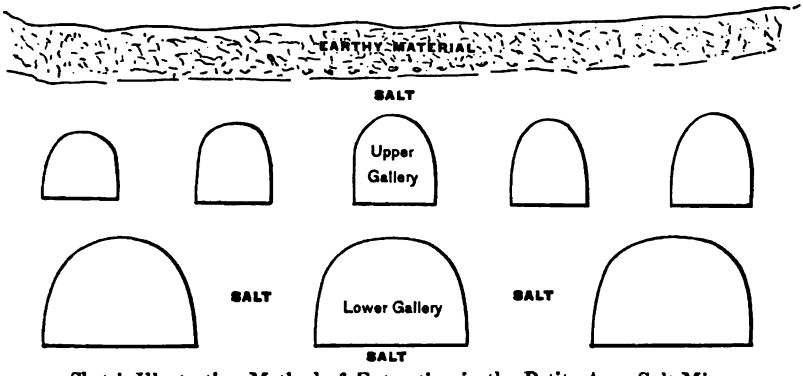
Analyses of Salt.

Varieties of.	Chloride of Sodium.	Chloride of Potassium.	Chloride of Calcium.	Chloride of Magnesium.	Sulphate of Lime.	Sulphate of Magnesia and Soda.	Alumina and Iron.	Residue.	Water.	Authorities.
Rock Salt.										
Avery Salt Mine.....	98.88	...	trace.	trace.	0.79	...	...	...	0.33	Goessmann.
Wieliczka, Austria...	100.00	....	....	trace.	...	...	...	...	...	Bischof.
Berchtesgaden	99.92	....	....	0.07	...	...	...	...	...	"
Hall, Tyrol.....	99.43	....	0.25	0.12	0.20	...	...	...	...	"
Hallstadt, Austria.....	98.14	trace.	....	....	1.86	...	...	...	...	"
Sta-sfurt, Prussia.....	94.57	...	....	0.97	0.89	...	1.12	2.23	0.22	Heine.
Vic, Ger. Lorraine.....	99.30	....	....	....	0.50	...	0.20	...	...	Berthier.
Jebel-Melah, Algeria	97.00	....	....	....	3.00	...	...	...	...	Fournet.
Ouled Kebbah, "	98.53	....	0.93	0.54	...	....	...	...	...	"
Carrick-fergus, Ire'd	96.28	....	....	...	3.50	0.08	...	...	0.14	G. H. Cook.
Santo Domingo.....	98.33	....	....	0.04	1.48	...	...	0.01	0.07	Goessmann.
Salt from Wells and Lakes.										
Onondaga, N. Y.....	97.41	...	0.15	0.18	1.26	...	...	...	1.00	G. H. Cook.
Pittsburgh, Pa.....	96.70	....	0.53	0.07	...	...	...	...	2.70	"
Kanawha, W. Va.....	91.31	....	1.26	0.43	...	...	...	...	7.00	"
Hocking Valley, O...	95.07	....	0.61	0.04	0.10	...	...	...	3.40	Goessmann.
Pomeroy, O.....	96.42	....	0.53	0.18	...	...	0.05	0.16	2.66	E. S. Wayne.
Cheshire, England....	96.36	....	0.01	0.02	1.17	...	...	...	2.44	G. H. Cook.
Dieuze, Ger. Lorraine	97.59	....	....	...	1.02	0.89	...	...	0.50	"
Droitwich, England	96.93	....	....	0.02	3.05	...	...	...	...	"
Golderich, Ont.....	97.03	....	0.01	0.03	1.43	...	...	...	1.50	Goessmann.
Saginaw, Mich.....	92.97	....	1.09	0.50	0.33	...	...	0.01	5.10	G. H. Cook.

The galleries of the second level are run 80 feet in width and 45 feet in height, leaving supporting pillars 60 feet in diameter. The lower pillars are so left that the weight of the upper ones rests upon them in part, if not wholly, with a thickness of at least 25 feet of salt-rock between.

There are 16 to 25 feet of earth above the salt-deposit. The contour of the latter conforms nearly with that of the surface. The working-shaft is 168 feet deep. The depth to the first level or floor is 90 feet; to the second, 70 feet further. The remaining 8 feet are used for a dump. The galleries of the first level were run, on an

FIG. 3.



Sketch Illustrating Method of Extraction in the Petite Anse Salt-Mine.

average, 40 feet in width and 25 feet and upwards in height, leaving supporting pillars 40 feet in diameter.

Fig. 3 is a rough draft of a vertical section. The galleries cross each other at right-angles and the ground-plan strongly resembles a chess-board.

Galleries aggregating nearly one mile in length have been run on the upper level and some 700 feet on the lower.

The stratification-plans are clearly denoted, and at first sight appear to have been thrown into a vertical position throughout; but a close examination reveals the fact that the structure is folded, and shows three anticlinal axes in the first level with the corresponding synclinals in the lower level. In *Harper's Monthly Magazine* for May, 1888, will be seen (p. 909) a cut of a salt-quarry at Iletsk, in which the structure is folded. The method of working shown is similar to that formerly employed at the Avery Salt-Mine.

In running a gallery the first work is the "undercutting" on the level of the floor, of sufficient height to enable the miners to work

with ease. The salt is then blasted down from the overhanging body. The yearly output is about 50,000 tons.

I am indebted to Major McCalla, the mining engineer of the company, for important data, and also for the following statement, which I give on his authority:

"The salt as it comes from the mine is dumped into corrugated cast-iron rolls which crush it. Next it goes into revolving screens which take out the coarser lumps for 'crushed salt' and let the fine stuff pass to the buhr-stones. These grind the salt, and from them it goes to the pneumatic separators, which take out the dust and separate the market salt into various grades. Taking the dust out is essential to the production of a salt that will not harden, since the fine particles of dust deliquesce readily and on drying cement the coarse particles together. The drill used in the mine is what is known as the 'Russian auger.' It is turned by hand and forced by a screw of 12 threads per inch. The holes take cartridges 1½ inches diameter. Two men will bore 75 feet of hole per working-day of 8 hours. Three-quarters of a pound of 18 per cent. dynamite is used per ton of salt mined."

Petite Anse Island is ten miles from New Iberia, with which it is connected by rail.

Unlike the barrenness that so often surrounds mines, the island is a beautiful park, with forests, grassy parterres, miniature lakes and streams. Sugar cane, cotton, corn, the Tabasco pepper, and a profusion of flowers grow here. The scenery is indescribably charming. The Teche is a lovely meandering bayou, the banks of which are fringed with live oak, cypress, pecan, ash and willow. It is the scene of busy industry. Along it are saw-mills, shingle-mills, brick-kilns, ice-factories, machine-shops, cotton-seed oil-mills, sugar-factories and cotton-gins. Cotton and other factories would certainly thrive here. Steamboats are constantly plying upon the bayou, some to New Orleans, some to Morgan City, and some between minor local points. The Teche is not a sluggish stream; its banks are high, and the country is never overflowed. The soil is excellent, and the climate healthy and delightful. The farmer, accustomed in the North to interruptions from snow and ice, can here work out of doors every week in the year. Invalids and others, worn out by the rigors of winter, can recuperate their wasted strength in this balmy atmosphere. The sea-breezes roll over the country and give health and long life to its inhabitants. The climate is, in short,

a medium between the tropical and north temperate, combining most of the advantages of both and the evils of neither.

To the geologist this is a country of constant surprises, in the form of materials brought from distant sources, and traces of ancient channels now abandoned. He finds clay from the Red River, alluvium from the Mississippi, and sand and gravel that remind him of the Bijou Basin in Colorado. Red River has been a fickle stream in its journeyings to the sea, changing its course from time to time, until finally it has lost much of itself in the waters of the Mississippi. But it has left its marks from the Sabine to the Atchafalaya.

NOTE ON A SPECIMEN OF GILSONITE FROM UINTAH COUNTY, UTAH.

BY R. W. RAYMOND, NEW YORK CITY.

(Buffalo Meeting, October, 1888.)

In connection with the paper of Mr. Locke on Gilsonite or Uintahite (*Trans.*, xvi., 162) read at the Salt Lake session of the Utah and Montana meeting, in July, 1887, the following data may be of interest and value.

In December, 1887, I received from Mr. William Wagner, Secretary of the Denver & Rio Grande Railway Co., a box of specimens of an asphaltic mineral from Utah. In the course of the correspondence which followed, I learned that the box had been forwarded by Mr. J. A. McMurtrie, of the firm of McMurtrie & Streeter, railroad contractors, Denver, Col. Mr. McMurtrie was, in 1883, chief engineer of the Denver & Rio Grande Railway, and the mineral was first brought to his attention in that year by Mr. Thomas H. Wigglesworth, who was then employed in making preliminary reconnoissances for wagon-roads to be used in constructing the Utah extension (now the Denver, Rio Grande & Western) of the railway. Mr. Wigglesworth, I understand, was subsequently chief engineer of the Colorado & Utah Midland, and is now again employed by the Rio Grande.

He writes substantially as follows concerning this mineral:

"The vein from which the box of asphalt sent you was taken, is about three miles east of Fort Duchesne, and near the line between the Uintah and Uncompahgre Indian reservations. It is from three to five feet thick, the difference in

thickness being apparently due to horizontal corrugations. The dip is at right-angles to the formation, which is about horizontal, and the course is N. 73° W. The vein can be seen for some distance in each direction cropping out on the ridges, where the wind has blown the sand into the valleys. The formation is a soft, dark sandstone, with intercalated layers of hard, dark-red clay. The asphalt cleaves from the walls readily and shows no signs of mixture with the sand or clay of the latter. The vein, where I saw it, had been opened to a depth of ten feet, and the material showed no change in character."

Mr. McMurtrie adds that a small quantity of the mineral has been mined and shipped to St. Louis, for the manufacture of asphaltic paint.

Through the kindness of Prof. T. M. Drown, I am enabled to present an analysis (I.) made in the laboratory of the Massachusetts Institute of Mineralogy, of the sample sent to me, to which I add, for comparison (II.) the analysis of Uintahite (Gilsonite) made in 1886 by Messrs. Fristol and Lawver, and given in Mr. Locke's paper, already cited.

	I.	II.
Carbon,	80.88	78.43
Hydrogen,	9.76	10.20
Nitrogen,	3.30	2.27
Oxygen,	6.05	8.70
Ash,	0.01	0.40
	100.00	100.00
	At 17° C.	Temperature not stated.
Specific gravity,	1.06237	1.065 to 1.07

These figures indicate clearly the same mineral, and the physical and other characteristics reported from the Massachusetts laboratory confirm this conclusion (as may clearly be seen upon comparing them with the corresponding data in Mr. Locke's paper):

"Very brittle; breaks with conchoidal fracture, and is easily powdered. Fuses at low red heat; burns with a brilliant flame; is capable of receiving impressions from a seal, like sealing-wax. Has considerable plasticity when warm, and on cooling presents a smooth, lustrous surface, and is brittle.

"Color, black, lustrous; streak and powder, dark brown.

"Soluble in carbon bisulphide and in benzol; partially soluble in oil of turpentine and in ether; slightly soluble in naphtha, and very slightly in alcohol."

I think it more than probable that this is not only the same mineral as that described by Mr. Locke, but that it comes from the same locality. Mr. Wigglesworth, however, has given me some interesting facts, indicating a wide distribution of the minerals of this group in the Utah region. He says:

"The same material occurs in several other localities. At a point north of White river, and some ten miles east of Green river, I saw it outcropping in an apparently vertical vein, thirteen inches thick. It occurs again in White river cañon, about twenty-five miles from Ouray, showing about the same characters and conditions, and also at a point in a side-gulch of the same cañon, near what is known as the Wagonhound, about thirty-five miles from Ouray.

"On the east side of the hog-back, between the Ashley and Uintah valleys, there is a substance which seems to be a mixture of liquid asphalt and sand. It is very hard and tough, and when broken, has a dark, oily appearance on the fracture. Where it has not been disturbed it lies in sheets. I have seen pieces of the same material north of White river and about twenty miles east of Ouray.

"The Ashley hog-back has been tilted to a dip of 40° S.W. Along its east face there is a thick stratum of soft sandstone, saturated with oil. The same stratum crops out on Green river where the stream cuts through the hog-back; but the oil seems to have seeped out.

"A material somewhat like that found near Fort Duchesne, but much harder and brighter, resembling jet, occurs on Strawberry creek, either as a stratum or oozing out between the strata, and hardening on exposure. The formation is a light-colored, thinly-bedded sandstone, dipping slightly to the east. This deposit can be traced several miles by the signs of burning on the sandstone. It has evidently at some time been on fire.

"In 1883, I found on a bar in Strawberry creek, a patch, about ten feet in diameter, of what looked like a soft asphalt pavement. On examination it proved to be very tough and pure. I supposed it to be the same material as that near Fort Duchesne—a piece that had fallen from some vein in the cañon, and had floated down, lodged on the bar and melted. Having since explored Strawberry cañon without finding any such vein, I am now inclined to think it had oozed out of the bed of the stream and hardened to the condition in which I saw it."

Mr. Wigglesworth says that he has heard of other localities where asphaltic minerals occur, but has no definite information concerning them. He is convinced that the supply of such material is practically inexhaustible in that region.

THE GLENMORE IRON ESTATE, GREENBRIER COUNTY, WEST VIRGINIA.

BY WILLIAM N. PAGE, POWELTON, W. VA.

(Buffalo Meeting, October, 1888.)

HAVING recently made a careful professional examination of this tract, I think its peculiar geographical, topographical and geological relations may prove interesting to members, not only because they are somewhat unusual in such a combination, but also as presenting the economical conditions of a region not widely known, at the western base of the Alleghanies.

Geographical.—This estate is situated in Greenbrier County, West Virginia, on Howard's and Anthony's creeks, tributaries of the Greenbrier river, and thus of the Ohio. It embraces about 7000 acres along "Bob's Ridge," covering, to a great extent, as shown by the accompanying map, Fig. 1, the valley on either side. Beginning within three miles of the Newport News and Mississippi Valley Railway (the old Chesapeake and Ohio), at the White Sulphur Springs, it extends northeast six miles to Anthony's creek, a large and reliable water-course which runs north of west to the Greenbrier, about three miles distant. Howard's creek, which occupies both valleys of Bob's Ridge, contains an ample water supply for any ordinary mining or manufacturing purpose. This stream makes comparatively broad valleys in the Hamilton shales and Helderberg limestones. Each branch heads against a small tributary of Anthony's creek, the water-sheds between being sufficiently low to admit railway crossings with easy gradients.

White Sulphur Springs is situated within a few miles of the Alleghany summit, from which there is all down-grade to the Ohio river; and, by following the Richmond and Alleghany Railway, down the James river, from Clifton Forge, a continuous descending grade has been established to the Atlantic seaboard. Pittsburgh, distant about two hundred miles, can be reached through present routes, either by the Ohio river, or the Baltimore and Ohio R. R., which crosses the N. N. & M. V. Railway at Staunton.

A company has been organized, and a charter obtained, for a railway to run from Lexington, Virginia, to Pittsburgh. Owing to the topographical features of the Cheat Mountain region, any such road would necessarily be forced to follow the Greenbrier river, which would give a direct line to Pittsburgh, from the vicinity of the Glenmore tract.

All of the streams above named afford easy gradients for a railway, and the turnpike from Dry creek to Alvon, though it leaves the creek, affords a practical route for a cheap line, as it is laid down on the map.

Topographical.—By inspecting the accompanying map, it will be seen that the Alleghany range here runs, as usual, from northeast to southwest. Between it and the river are two parallel ranges of practically the same altitude; but, while the Alleghany is nowhere cut between long limits, these parallel ranges are frequently so. Greenbrier mountain is cut to its base by Anthony's and Howard's creeks, and another small stream between; while the central range, or

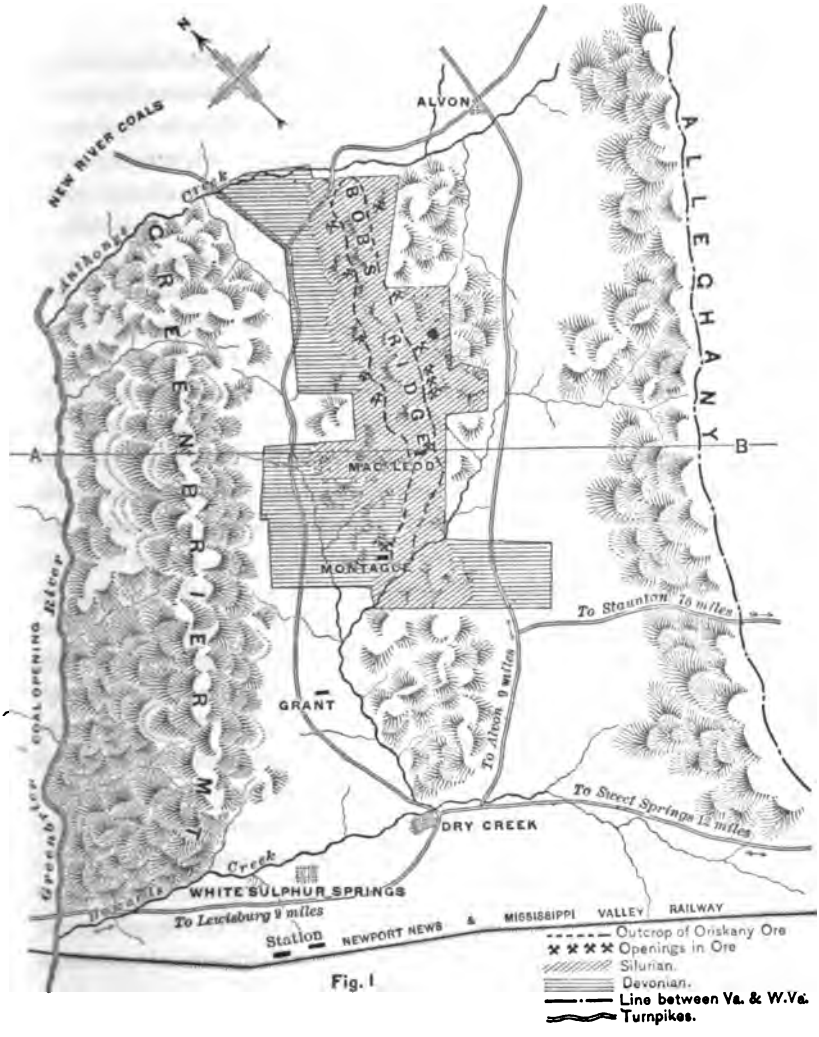


Fig. 1

GEOLOGICAL & TOPOGRAPHICAL
Map of Part of
GREENBRIER COUNTY, W.VA.
Showing
THE GLENMORE ESTATE.
Scale 1 inch — 3 Miles.

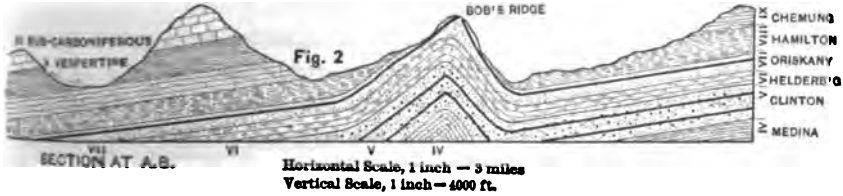


Fig. 2

Horizontal Scale, 1 inch — 3 miles
Vertical Scale, 1 inch — 4000 ft.

"Bob's Ridge," is also cut by Dry creek, as well as the above-named streams. All these gaps afford easy routes from one valley to the other, and there are no obstructive cliffs or waterfalls to hinder any such passage. Except where the limestones are cut, the slopes are smooth and regular, with a succession of spurs and small ravines on every slope. The Alleghany, being the Appalachian water-shed, is cut by tunnels only; but its slope bears a close resemblance to the others. The Newport News and Mississippi Valley Railroad crosses it within less than four miles by a tunnel over four thousand feet in length, at an altitude of two thousand feet above tide.

Geological.—The summit of the Alleghany occupies about the central line of pressure which elevated this section of the Appalachians. The base of such a force must have been many miles broad, since all foot-hills are parallel ranges of nearly equal altitudes and strata-disturbances. The main water-shed is sometimes anticlinal, but at this point it is a monocline, tending to a synclinal axis. To the eastward the strata have been broken into series of axes, and little is left above the Devonian—the Medina sandstones, as a rule, occupying the summits. To the westward, however, Bob's Ridge is the only anticlinal axis, the Greenbrier mountain being a monocline, with the Subcarboniferous on top. Immediately west of this point the heavy ledges of the Carboniferous (which lap up to the Greenbrier) have resisted the force so effectually that not a single break or fault has been found in the West Virginia coal-field, the strata of which have nearly a uniform dip to the northwest of about one hundred feet to the mile, until the synclinal basin of the Ohio is approached. Some of these ledges are more than two hundred and fifty feet of massive sandstone, and the Carboniferous develops a thickness of more than two thousand feet before reaching the Ohio. The force has been sufficient to tilt this enormous mass, which refused to bend within ninety or one hundred miles. By reference to the accompanying section, Fig. 2, it will be seen that Bob's Ridge occupied a line of the greatest pressure, since it was the fulcrum against which this entire force was sustained, the strata on neither side giving way to afford relief. It will also be seen that the Silurian, of which the Oriskany (VII.) forms the uppermost measure, is here on the surface for the last time westward. The Devonian shales are largely developed on both sides of this ridge, and constitute the principal valley-formation, though in places, near the southwest extremity of the property, Howard's creek has cut through the Helderberg, which leaves massive cliffs of a remarkably pure limestone. This forma-

tion is here about three hundred feet thick, and is almost entirely a variegated marble, resembling that of Tennessee, which comes from the same formation.

The Oriskany, lying immediately on these limestones, is one of the great iron-ore-bearing formations of Virginia. The Vespertine (as classified by Rogers), lying under the Greenbrier limestones, forms, with them, the Subcarboniferous. The Catskill is entirely missing. The Vespertine is a red shale of peculiar structure, and carries a regularly stratified vein of carbonate iron-ore, from three to five feet thick, of which both Prof. Rogers and Prof. Dana,* especially the former, speak in favorable terms. Rogers traced this stratum for thirty miles, and, from his numerous analyses, says it is one of the most valuable iron-deposits in the world. I have seen it within a stone's throw of the mouth of Howard's creek, but never had it analyzed. A seam of coal has been opened in this formation, within a few feet of this ore; about two miles up the river, marked "Coal Opening" on map, which is from six to eight feet, and the regular New River coals are found a few miles further up to the northwest. These coals are cut by the Greenbrier river in the adjoining county of Pocahontas, which is the eastern escarpment of the West Virginia coal area, covering sixteen thousand square miles. Thirteen workable measures have been proved, aggregating about seventy feet of coal.†

Ores.—A persistent seam of iron-ore has been traced and proved by numerous openings and cuttings through a distance of six miles on both sides of Bob's Ridge, making a total outcrop of about twelve miles, as approximately located on the map by crossed hammers along the dotted outcrop line. This ore-measure is perfectly regular, conforms to the under- and over-lying strata, and has an exposed thickness nowhere less than eight, and in places more than fifty feet, measured at right angles to the line of dip. As shown on the section, it blankets the surface on large areas, which can be quarried with comparatively little stripping. I have seen no ores in Virginia, Alabama or Tennessee, which, in my opinion, can be mined under greater advantages, or at less cost. The great value of this ore, however, is its peculiarity of structure and quality. It is uniformly, throughout the estate, an open-structured red oxide, lying in massive ledges. Numerous analyses give from seven to

* See Rogers's *Geology of the Virginias*, edition of 1884, pp. 105 to 108, 135, 161, 162, 163, 185 and 201; and Dana's *Geology*, 3d edition, p. 293.

† See my section of Hawk's Nest, published in *The Virginias*, vol. i., No. 2, p. 22, and map.

nine per cent. of total water. It is so highly oxidized that the hands become reddened from the touch, and men are dyed a bright red in working it. The following is a fair analysis, made by C. B. Manby, Chemist to the National Tube Works, of McKeesport, on samples taken thirty feet below surface:

	Per Cent.
Ferric oxide,	87.15
Siliceous residue,	1.98
Manganic oxide,	*trace.*
Water (total),	8.50
Phosphoric acid,	.12
Sulphuric acid,	.10
Alumina, Lime, Magnesia, } Principally Alumina,	2.15
Total,	100.00

Metallic iron, 61 per cent. Phosphorus, 0.07 per cent. Sulphur, 0.05 per cent.

Nowhere else in the Appalachian has a similar ore been discovered in workable quantities, to my knowledge; and though found here in the same formation as elsewhere, the geological conditions have been such as to render it anhydrous to a great extent. The conditions previously stated would necessarily result in extraordinary pressure and its consequent heats. That some such powerful oxidizing agency has been at work here is evidenced by the sandstones and other rocks, as well as this ore. Wherever iron occurs, even as coloring or streaks, it is in the shape of red oxide; and the Oriskany sandstones are nearly metamorphosed, as compared to those of the East. That this oxidation is not merely a surface-effect is amply demonstrated by such conditions, and in some places the ore has been driven thirty feet under cover without change. East of the Alleghanies, the Oriskany sandstones are largely developed, and the ore is frequently replaced by them; but here these sandstones are much less prominent, and are cherty. The small percentage of silica found in the ore is evidence that the sandstones do not predominate; and it may be accepted as a fact that wherever the one is undeveloped, the other attains its maximum thickness and regularity; the rocks here being just on the border of perfect regularity, are less disturbed and more uniform than I have seen elsewhere in this formation.

Limestones.—The Helderberg limestone, containing about 98 per cent. carbonate of lime, is so cut by Howard's creek, that convenient

quarries can be located along cliffs of over two hundred feet of massive limestone. This stone is probably one of the best furnace-fluxes in America.

In addition to this, the Subcarboniferous (XI.) is an excellent limestone, largely developed over the greater part of Greenbrier County, and is the soil-base of its blue-grass section. This limestone is in position near the mouth of both Howard's and Anthony's creeks, as well as along both sides of the Greenbrier between.

Timber.—Large quantities of white oak and other hard woods cover the entire acreage of this estate. The white pine has been largely cut off, but enough remains for all building and construction purposes. The adjoining lands to the northeast are covered with a dense growth of this timber, which may be floated down Anthony's creek. This property, unlike most of the ore-lands east of the mountain, has never been cut for charcoal, since the day of charcoal operations in this section had passed before the opening of any trans-Alleghany route.

Coal.—As previously stated, a seam of coal has been opened near the bank of Greenbrier river, about one hundred feet above water-level, and worked, to a considerable extent, as fuel for the town of Lewisburg, eight miles distant. This seam lies nearly horizontal, and will average about six feet in thickness. Its position is shown on the map at the point marked "Coal Opening." An analysis of this coal, recently made by Mr. Ford, of the Messrs. Carnegie Bros. & Co., Ltd., gives the following results:

	Per cent.
Carbon,	82.162
Sulphur,	.900
Volatile matter,	5.050
Ash,	11 808
Total,	99 920

The limits of this deposit are unknown, but the evidences point to an extensive area. Lying, as it does, in the Vespertine, I am unwilling to place much confidence in it, either as regards quantity or quality, but must admit that an inspection of the old workings is very favorable to both. As will be seen from the analysis, it is graphitic, another evidence of unusual pressure in this vicinity. It is sufficiently hard and compact to use in the furnace, without preparation. Apart from this coal, however, the regular New River coking measures are within such easy reach, that their hills are within sight from the ore-outcrop on the northwest slope of "Bob's Ridge."

These New River coals are worked very extensively along the Newport News and Mississippi Valley railway, within fifty miles of White Sulphur Springs. This railway follows the Greenbrier southwest, about forty miles, to its junction with New river, at Hinton, through a limestone valley, before entering the regular coal-measures, whereas, a few miles up the river from Anthony's creek, these measures can be reached nearer water-level. Coal has also been opened north of Anthony's creek, near this tract, but I have not examined it. It opens several feet thick; but I take it to be the Vespertine, from its general position. After passing the head of Anthony's creek (which is thirty or forty miles in length), the Bob's Ridge axis disappears, and the coals lap on the main Alleghany mountain. The continuation of Bob's Ridge, through this distance, is known as Beaver Lick mountain, on either side of which is a valley of the creek named, similar to those of Howard's creek, but more extensive.

Conclusion.—The condition of this estate may be safely summed up as follows, viz.: Nearly the whole seven thousand acres are underlain near the surface by a regular measure of red ore, averaging, probably, more than thirty feet in thickness, containing, at least, sixty per cent. of metallic iron, of such structure that comparatively little fuel or flux will be required to reduce it, and remarkably low in phosphorus, sulphur and other impurities. If not a Bessemer ore, alone, it can be largely used as a mixture for such. It is of inestimable value as a mixture for the neighboring Virginia ores, containing from three-fourths to one per cent. of phosphorus, making neutral what are now cold-short irons, and reducing fuel, flux, and labor. From three to nine miles of cheaply constructed railway, with comparatively level gradients, will develop both ores and limestones; and coals of long-established metallurgical reputation are within sight.

An abundant water-supply can be relied upon at all seasons, and the valleys afford ample room for every undertaking. Gravity machinery is all that will be required for mining and drainage. There is an open market for every ton of ore to be mined, over present routes of transportation, and, though additional lines will add value, they are not necessary.

As adjuncts to a furnace-site, fuel, ore, flux and water are all in sight, and within a radius of ten miles. Such conditions of quantity and quality exist nowhere else in America. It is hundreds of miles nearer the market than Alabama, and railway freights, which consti-

tute such a large proportion of the cost of iron, are here reduced to a minimum. Its position is in a moderate climate, and one of the healthiest spots in the world. Labor is cheap and plentiful. Ten miles of haul will cover all material necessary for the production of pig-iron, which will be within two hundred miles of Pittsburgh, and about the same distance from the Atlantic seaboard. Ninety miles of down-grade, over the N. N. & M. V. railway, will reach the navigable waters of the Ohio, and open all of its markets to an article which can have no competition short of the Lakes.

The ores can be delivered on cars at $2.00 per ton, or 3½ cents per unit of iron, and pay on mining investment; limestone at 75 cents, and coke for not more than $2.50. A fair cost for producing a ton of pig-iron, under these conditions, would be about as follows, viz.:

Ore at 3½ cents per unit,	$3.33	
Limestone, ⅔ tons, at 75 cents,	50	
Coke, 1 1/5 tons, at $2.50,	3.00	
Labor, per ton of iron,	2.00	
Incidentals, per ton of iron,	1.00	
Salaries, etc., per ton of iron,	1.00	
	10.33	—Cost at furnace.
Present rate on iron, to Pittsburgh,	3.75	—All rail.
Cost in Pittsburgh,	$14.58	

Taking into consideration the character and structure of the ore, every other item is over-estimated, and this cost is not such an arbitrary assumption as is generally made by parties having no practical experience. Having had the experience, I feel safe in saying that the above estimate is more than liberal. Add to this estimate the proportional railway charges on each material, and the cost will exceed that claimed by any furnace in the South, and probably Pennsylvania as well. The Virginia furnaces pay from $1.50 per ton upwards as freight on fuel, and those that buy ores, from 50 to 75 cents freight, on a grade rarely running over forty per cent. Assuming the fuel to be the same (which, of course, is not true between a forty and a sixty per cent. ore), there will be added on coke $1.80, and taking the ore at 60 cents (a low rate), $1.50 more will go into freights. Add to this 20 cents only for flux, leaving labor and incidentals the same, the cost at furnace will be $14.33, probably a little more than the present cost at Low Moor, though considerably more than that claimed. It must be borne in mind, in making these comparisons, that one iron will contain one per cent.

of phosphorus, or more, while the other will contain only about one-tenth as much, or less. With a direct line to Pittsburgh, the freight to that point will be reduced to about $1.50, and $1.00 should cover the present charges to the Ohio, since all rolling stock going in that direction is comparatively empty. Both the Low Moor and Longdale furnaces claim to produce a ton of iron on less coke than I have named, though their ores are much leaner, and, comparatively, highly siliceous, requiring a very large percentage of limestone. At one of these furnaces it is also claimed that labor has been reduced to 90 cents per ton.

ANTHRACITE AND COKE, SEPARATE AND MIXED, IN THE WARWICK BLAST-FURNACE.

BY EDGAR S. COOK, POTTSTOWN, PA.

(Buffalo Meeting, October, 1888.)

THE Warwick furnace at Pottstown, Pa., constructed for anthracite fuel, is, as may be remembered, 55½ feet high, with 15½ feet bosh. The actual working height from stock-line to bottom is only 47½ feet, the bell-and-hopper taking up 8 feet.

During the year 1886 we used mixed fuel, in the proportion of three-fourths anthracite and one-fourth coke. The total product of iron was 30,629½ tons of 2280 lbs., being a weekly average of 589 tons, including all ordinary and extraordinary stoppages. Deducting the latter, the weekly average would be about 600 tons. The fuel-consumption for the year was 1.2 tons per ton of iron, while the ore-mixture yielded 52 per cent. of iron.

For about two months during the summer of 1887 we were obliged to run on anthracite exclusively, owing to the Connellsville coke strike.

In the fall of 1885, in relining the furnace, the crucible had been built 8 feet 9 inches in diameter, with the view of using regularly one-quarter of coke (the largest previous diameter having been 7 feet 5 inches), and we felt somewhat apprehensive of the results when compelled to use anthracite only. The average of seven weeks' run on coal, as compared with the average of the seven corresponding weeks of 1886 (using one-quarter coke), was as follows:

Date.	Fuel.	Iron made per week. Tons.	Fuel per ton of iron. Tons.	Ore. Per ct.	Limestone per ton of iron. Tons.
1887, seven weeks, .	coal.	523	1.262	52	0.7250
1886, seven weeks, .	¾ coal, ¼ coke.	556	1.278	51.3	0.7125

Date.	Exposure of stock Hours.	Burden.	Air per minute. Cu. ft.	Pressure. Lbs.	Heat of blast. Degrees F.
1887, seven weeks,	20¾	1 to 1.55	10,907	8.2	938
1886, seven weeks,	18½	1 to 1.55	10,978	7.6	917

A difference of only 33 tons iron per week is shown in favor of mixed fuel; but the poorest work of the blast thus far was made in the summer of 1886. The corresponding weeks of 1888, three-quarters coal and one-quarter coke, show as follows:

Date.	Fuel.	Iron made per week. Tons.	Fuel per ton of iron. Tons.	Ore. Per ct.	Limestone per ton of iron. Tons.
1888, seven weeks, .	¾ coal, ¼ coke.	634	1.1375	54	0.65

Date.	Exposure of stock. Hours.	Burden.	Air per minute. Cu. ft.	Pressure. Lbs.	Heat of blast. Degrees F.
1888, seven weeks,	17.6	1 to 1.65	11,048	6.4	929

In making comparisons it is only fair to compare corresponding weeks of each season, for the work of the furnace varies materially with the season of the year. While the results obtained in 1886 and 1887 do not differ greatly, the management was much more arduous in 1887. The furnace was exceedingly troublesome to control—far more so than with mixed fuel. The "ore-dirt troubles," of which mention has been made previously in the papers of Mr. John Birkinbine and discussions following, were more frequent and more dangerous, the dirt-slips frequently coming without a moment's warning. In the seven weeks we had two narrow escapes from possibly serious consequences. But for the constant fear of "dirt-troubles" more iron could have been made with somewhat lower fuel-consumption. We were glad when the coke strike terminated.

During the fall of 1887, and until the close of the year, the weekly product (using one-fourth coke) averaged 625 tons; and the year was ended with a total make of 30,660½ tons, just 31 tons in excess of 1886.

No sooner had we safely passed through the short strike on the Reading Railroad, and, as we supposed, become comfortably settled

for the winter, than rumors of labor troubles in the Schuylkill coal regions became prevalent, and were quickly followed by an almost total suspension of mining. One individual colliery, from which we obtained about one-third of our usual supply, continued in operation. But the men would not permit the coal mined to be loaded into Philadelphia and Reading cars; and as the Pennsylvania Railroad could not, or would not, furnish cars to carry it over their Schuylkill Valley Branch, our stock of coal was soon exhausted. About the middle of January, 1888, we commenced using coke only —a mixture of Connellsville and Clearfield. Being unable, for want of track- and storage-room, to keep the two varieties separate, we soon dropped Clearfield, because we found it impossible to fill the furnace uniformly when using the two cokes hap-hazard, just as they happened to be unloaded. Had it been possible to fill a definite and fixed proportion of each, this trouble would not have been experienced, whatever might have been the economical results.

The first week's work with coke only was disappointing. The furnace began to scour, and continued doing so for a week. To keep the crucible hot, it was found necessary to reduce the burden, which had been 1.75 ore to the pound of coal and coke, to 1.5 pounds ore to the pound of coke. The iron was thus prevented from falling below gray forge in grade; and the slag, though black, was very hot and liquid. Our bosh-coils and jacket became hot. For a while it looked as if we would lose the former; but, by close watching and frequent application of water from a powerful pump by means of hose directly to the coils, they were saved. Doubtless a large portion of the brick-work of the bosh had been melted out months before and replaced with scaffold-material. The first action of the coke was to remove this filling. After the scouring ceased the furnace became much hotter and grayer, and the burden was rapidly increased to 1.7 pounds ore to 1 pound coke.

Our first receipts of coke were of average quality. But "it's an ill wind that blows no one any good." The coal-strike gave our Connellsville friends an opportunity to dispose of their stocked coke. If all the eastern furnaces fared as did Warwick, they received and paid for their fuel in about the proportion of one-fourth dirt and three-fourths coke. The stocked coke was also completely water-soaked, so that it was impossible to thoroughly separate the dirt; and, for the most part, it had to be filled as received. As the shipments of the dirty water-soaked coke increased, both the

burden and the yield were reduced again, until 560 to 580 tons of iron per week became the usual work. The following shows the average results obtained from the use of coke only, from January to March, 1888, compared with the corresponding weeks of 1877, using three-fourths coal and one-fourth coke:

Date. January to March.	Fuel.	Iron made per week. Tons.	Fuel per ton of iron. Tons.	Ore. Per ct.	Limestone per ton of iron. Tons.
1887, . . .	¾ coal, ¼ coke.	615½	1.15	50.9	0.65
1888, . . .	all coke.	570	1.22	50.5	0.79

Date.	Exposure of stock. Hours.	Burden.	Air per minute. Cu. ft.	Pressure. Lbs.	Heat of blast. Degrees F.
1887, . . .	17	1 to 1.75	11,118	6.4	944
1888, . . .	14	1 to 1.62	10,485	5.2	934

The quality of the iron made was in favor of the mixed fuel, as shown by repeated analyses. The silicon and sulphur in the coke-iron were considerably higher than usual, the phosphorus remaining about the same:

	Silicon. Per ct.	Phosphorus. Per ct.	Sulphur. Per ct.
Coke-iron, average, . . .	0.945	0.352	0.101
Mixed-fuel iron, average (8 mos.),	0.588	0.296	0.037

It was found impossible to melt as many tons of ore per week with coke only as we had become accustomed to melt with three-fourths coal and one-fourth coke. The furnace-stack being low (only 47½ feet effective working height), and the coke so much more bulky than coal, the same rate of driving emptied the furnace in from eleven to twelve hours. For several months previous to using coke only, the stock had passed through the furnace in from fourteen to fifteen hours. This seems to be our economical limit, except under temporary favorable conditions. Eleven to twelve hours was found to be too short a period for the economical reduction of the ores; and it became necessary to slacken the speed of the engine and prolong the exposure of the ore to the action of the gases. We hoped that the coke-gases on the smaller weight of ore in the furnace, would act more energetically and quickly than the gases from coal, thus permitting a shorter exposure; but in this we were greatly disappointed. The yield of ore-mixture and the burden being the same, slower driving was synonymous with reduced output of furnace. As the average burden with coke only was lower than the burden car-

ried by mixed fuel, the enforced experiment of using coke exclusively was anything but a financial success. With a higher stack the results would have been more favorable. Both as regards yield and economy we had been obtaining maximum results from our little furnace considering the ores used, and we could do no more with coke only; but, on the contrary, did worse than with three-fourths coal and one-fourth coke. Had we been averaging from 350 to 450 tons iron per week, instead of over 600 tons, and had we then, by using coke, increased the yield to 570 tons per week, no doubt we should have considered the increase very satisfactory, and thought that it partially repaid the extra cost of the coke as compared with the price of coal.

During the two months of the coke-experiment our old enemy, "ore-dirt troubles," practically disappeared; the furnace was more easily managed, working smoothly and regularly. Shortly after the stocked coke, with all its dirt, began to arrive in increased quantities, we became acquainted with a distant relative of the old enemy. The ore-dirt slips gave place to slips of coke-dirt sufficient to thicken the slag and necessitate the removal of the 2-inch cinder-notch, in order to get the slag to flow freely. Though annoying at first, these troubles were not dangerous; and, after substituting a 3-inch cinder-notch, we had no particular difficulty. Had our stack been 70 feet high, no doubt our trial of coke only would have been eminently satisfactory from a technical point of view. Of course, the difference of cost between it and the usual price of coal is too great to be equalized by any possible improvement in working.

From these two trials of all coal and all coke, as compared with a mixture of three-fourths coal and one-fourth coke, it appears, other things being equal, that the latter gives better results in a furnace with comparatively large crucible and low stack, than either of the two used alone. The mixture of one-fourth coke assists in the driving of an anthracite furnace, while the coal, in a low stack, will carry more burden and permit more ore to be melted than will coke alone. Under different conditions, that is, with high stack, all coke, compared with a mixture of the two fuels, would doubtless give better results, especially when the greater difficulties attending the management of an anthracite furnace are taken into consideration.

The change from all coke to mixed fuel in March gave rise to irregular workings, the reverse of those experienced in January. The bosh of the furnace, enlarged from the continued use of coke,

proved to be too large for a mixture of three-fourths coal and one-fourth coke. There was a constant tendency on the part of the furnace to form new lines to conform to the changed conditions. Instead of filling up slowly, and reducing the area gradually, the disposition was to fill rapidly and too much at once, indicated by casts much too small for the amount of stock filled. This filling, being soft and pasty, would cause stock to settle slowly along the walls, and frequently to stick fast, and occasionally to arch over and refuse to settle at all. For several weeks it was a continuous struggle to keep the furnace in successful operation. The constant tendency to stick and jump, to fill up and tear away again, kept the manager in anything but a happy and contented frame of mind. The amount of iron made for the month was fairly satisfactory—2605 tons—but the fuel consumption reached 1.204 tons, with ores averaging 51 per cent. Towards the latter part of March the bosh of the furnace got into better shape—better proportioned to the fuel used. Bosh-coils and plates cooled off considerably, the bosh-filling became hardened, wearing smoother, and the stock settled regularly and evenly. So we began April under favorable conditions, and made for that month the best record in the history of the furnace, reaching a total tonnage of 2792½ tons, an average of 93.8 tons per day, with a 51 per cent. ore-mixture and 1.16 tons fuel per ton of iron. The largest weekly yield was 687½ tons, and the largest daily product 110½ tons. This work was continued through May, with a product of 2809½ tons, the fuel-consumption being 1.13 tons per ton of iron, and the ores yielding 52.5 per cent. The May product was reduced by about 50 tons, owing to a serious outbreak of iron, setting fire to the cast-house and doing other damage, but not sufficient to put the furnace out of blast.

THE FLUE-DUST OF THE FURNACES AT LOW MOOR, VA.

BY ELLISON C. MEANS, LOW MOOR, VA.

(Buffalo Meeting, October, 1888.)

MANY Virginia furnaces are troubled with a small percentage of zinc in their stock, this element being present, although the chemist may have failed to report it, either in the ore or in the limestone.

When many hundreds of tons are passed through the furnace, the zinc appears in the flues to a marked degree. Sometimes the quantity is such that the metal appears through the walls of the furnace; but this a rare occurrence.

At Low Moor, when Furnace A was blown out, at the completion of blast No. 3, a mass of high-grade zinc oxide, weighing nearly six tons, was found suspended to the lining, almost directly below the tunnel-head. The presence of this mass and its effect on the charging, would readily explain the bad working of the furnace on one side.

The tunnel-head itself had been a source of constant care during the blast. The zinc would accumulate there in such quantities as to affect very materially the passage of gas into the down-take, and thus cause a strong back-pressure on the blowing-engines. There seemed to be no remedy for this but to take the blast off the furnace and knock out the deposit. This enlarging of the tunnel-head became necessary once a month. Probably the only way to reduce this deposit would be to construct the down-take so as to enter the furnace at as small an angle as possible, and thereby avoid any bench-room for a deposit at that place.

The introduction of a dust-catcher with a small bell has greatly reduced the amount of dust in the flues. Every day this bell is lowered at least twice, and the dust is allowed to fall out from above. The amount of dust thus obtained is usually about 600 to 1000 pounds a day. The analysis, by J. C. Smith, in the laboratory of the Massachusetts Institute of Technology, of a sample from the dust collected at this point shows it to have the following composition:

	Per cent.		
Zinc oxide,	87.660		
Protoxide of iron,	1.270		
Binoxide of manganese (MnO_2),	3.600		
Alumina,	0.228		
Lime,	1.850	Metallic zinc,	70.356
Silica,	2.410	Metallic iron,	0.988
Phosphoric acid (P_2O_5),	0.701		
Sulphuric acid (SO_3),	0.493		
Loss on ignition,	0.640		
	98.852		

The deficiency in the total of the above analysis was ascribed by Mr. Smith to his omitting to determine magnesia and alkalies.

Possibly there may have been a loss of some zinc. At all events, I obtained by a volumetric method a trifle over 71 per cent. of zinc from the same material.

The analysis of a sample taken from the flue in front of the boilers, one hundred and fifty feet from the furnace, shows about 30 per cent. of metallic zinc in the form of oxide. The mass found inside of the furnace is almost pure zinc oxide.

Furnace B, at present in blast for the first time, being five feet higher than Furnace A, will probably, when blown out, be found to contain a larger mass beneath the tunnel-head than did Furnace A.

NOTE ON CAST-STEEL WATER-JACKETS.

BY RICHARD H. TERHUNE, SALT LAKE CITY.

(Buffalo Meeting, October, 1888.)

THE use of water-cooled breast-jackets or cinder-tap blocks is a great convenience in lead-smelting, even when siliceous slags are made. If the charges are at all basic it is almost imperative. Jackets of riveted wrought-iron or steel are expensive; those of cast-iron are liable to crack; those of bronze are costly and not durable, being easily penetrated by tapping-bars or destroyed by streams of matte impinging against them. To overcome these objections I introduced at the Hanauer works, in April, 1886, cast-jackets of steel, which have been in constant use since that time and are to-day as good as new.

These jackets were made for us by the Chester Steel Castings Company, of Chester, Pa., at a cost of ten cents a pound at the works of that company. The metal is $\frac{5}{8}$-inch thick, and the jackets were cored and tapped for $1\frac{1}{2}$-inch bushings, affording two large holes for the removal of dirt and incrustations. The same holes are used for feed- and discharge-water through $\frac{3}{4}$-inch pipe.

I hope the successful introduction here of cast-jackets of steel will lead to their more extended use in all branches of metallurgy where water-cooled surfaces are severely taxed, and where the size and thickness of the castings are not unfavorable to the production of a sound article, impervious to water and of high tensile strength.

A DIFFERENTIAL REGENERATIVE HOT-BLAST STOVE AND ITS APPLICATION TO AN OPEN-HEARTH BLAST-FURNACE.

BY JACOB T. WAINWRIGHT, PITTSBURGH, PA.

(Buffalo Meeting, October, 1888.)

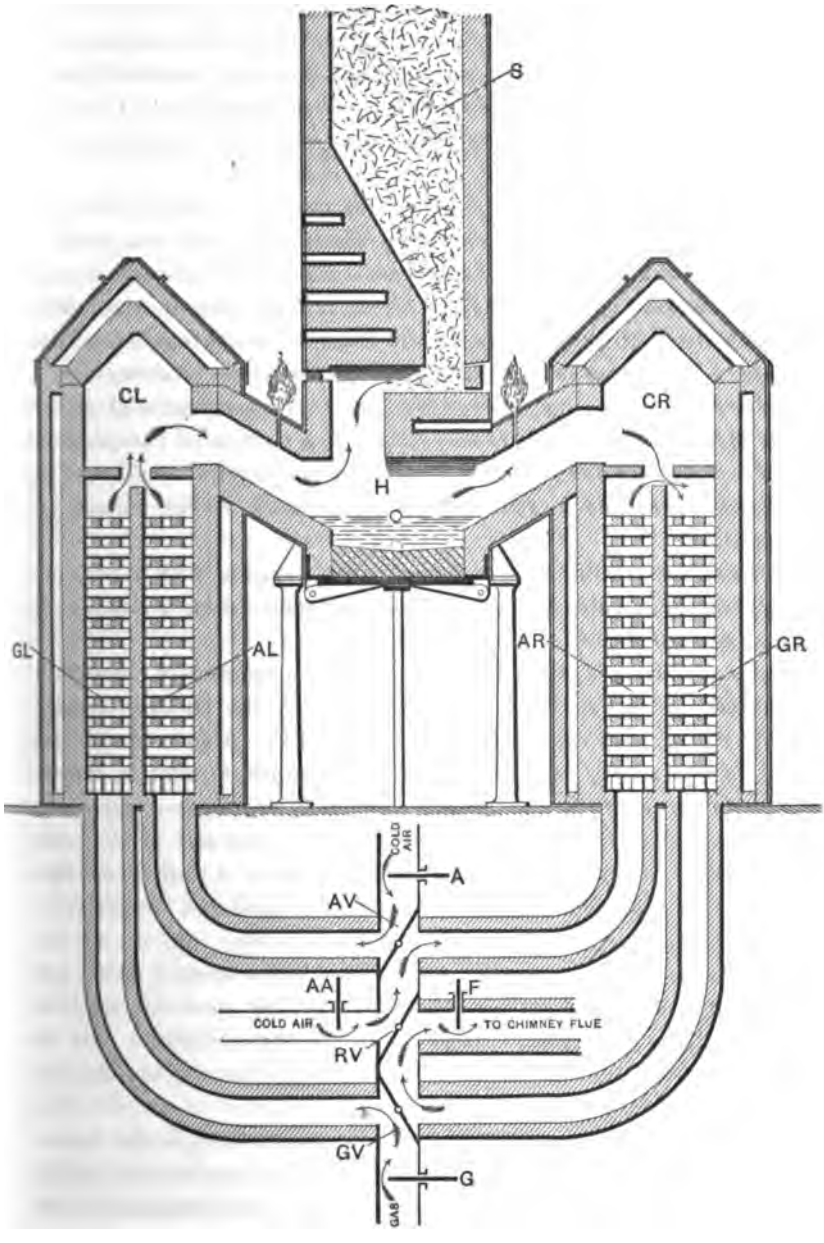
THIS stove has been designed to meet the requirements of a furnace that must be operated with either a reducing or a neutral flame; and more particularly to make feasible the operating of reduction-furnaces with the cheaper grades of gas-fuel, which has heretofore been impracticable.

In general, cheap gas-fuels result from incomplete combustion, and, being suited for further combustion, answer very well for heating purposes; and high temperatures may be attained with them, when they are used in conjunction with regenerative stoves.

But, in operating a reduction-furnace, a high temperature is not the only requirement. It is also necessary that the fuel shall not be consumed beyond a certain degree, in order to insure a reducing flame for the reduction-process; consequently, it becomes necessary to dispense with combustion as much as possible previous to the use of the fuel-gases in the reduction-furnace. Moreover, these gases must be preheated to a high degree, in order that a sufficient temperature may be maintained in the furnace; and, since this preheating cannot be accomplished by a previous, partial combustion of the gases themselves, it follows that the stoves for preheating these gases should be heated to as high degree as it is possible to attain with an inferior fuel. This result can best be accomplished by *complete* combustion of the heating gases, and is further facilitated by preheating *all* of the air and gas used for this complete combustion. It is more particularly desirable to preheat the air, since the combustible gases used may generally be obtained directly from the furnace, or gas-producer.

The reducing gases, escaping from a gas-fuel reduction-furnace, are capable of considerable further dilution with air to secure complete combustion, and consequently may be utilized for heating the stove to a high temperature, provided the air be sufficiently preheated.

Fig. I.



DIFFERENTIAL HOT BLAST STOVE.

In the well-known Siemens type of regenerative furnace, which preheats the gas and air in separate checker-work, this requirement can be met by intercepting the flues from the usual reversing-valves to the chimney-flue, with an auxiliary four-way reversing-valve, connected with an air-flue opposite to the chimney-flue. This valve should be reversed more frequently than the other reversing-valves, and independently of them.

The effect of this arrangement is to cut off alternately from the chimney-flue the two regenerative chambers which are being reheated, and at the same time to pass a current of air back through the intercepted chamber, thereby highly heating this air, and causing it to unite with the gases from the furnace, pass through the alternate of the intercepted chamber, and thence to the chimney-flue.

The accompanying illustration shows in a conventional manner an open-hearth blast-furnace, in which a portion of the preheated gases is passed through the burden in the vertical shaft, and the remainder is used to reheat the checker-work in the manner just described.

The object of this type of furnace is to dispense with a large proportion of solid fuel in the vertical shaft, and utilize a cheaper gas-fuel as a substitute.

In operation, a blast of cold air passes continuously through the damper, *A*, and is directed upward through the checker-chamber, *AL*, by means of the reversing air-valve, *AV*. Also, a blast of gas-fuel continuously passes through the damper, *G*, and is directed upward through the checker-chamber, *GL*, by means of the reversing gas-valve, *GV*. These currents then combine, and produce combustion in the chamber, *CL*; and the dampers, *A* and *G*, are regulated to admit the desired proportions of air and gas, respectively.

The gases pass from this combustion-chamber into the furnace-hearth, *H*, whence a portion of the gases passes upward through the burden in the shaft, and performs the smelting operation, while the remainder of the gases passes from the furnace-hearth into the chamber, *CR*, and thence downward through the checker-chamber, *GR*.

Also, a continuous blast of cold air passes through the damper, *AA*, and is directed upward through the checker-chamber, *AR*, by means of the auxiliary reversing-valve, *RV*, and thence downward through the checker-chamber, *GR*, burning the gases from the chamber, *CR*; and the damper, *AA*, is regulated to admit the desired amount of air for complete combustion. From *GR* the gases

are directed through the damper, F, into the chimney-flue, by means of the reversing-valves, GV and RV; and the damper, F, is regulated to pass the desired proportion of gases through the vertical shaft, S, and the chamber, CR, respectively.

The result of reversing the valve, RV, is to alternate the action in the chambers, AR and GR, and does not affect the gases that pass *into* the furnace. The result of this alternate action is to maintain these regenerative chambers at an extremely high temperature in the upper part, and at a comparatively cool temperature in the lower part, thus forming a most efficient stove.

The valves, AV and GV, are reversed together in the usual manner, for the purpose of alternating the action in the stoves, CL and CR, and thus maintaining the regenerative action upon the gases that are passed *through* the furnace. Uniform results are insured by regularity in reversing these valves.

It may be mentioned as a desirable feature, that air, thus passed periodically through the checker-work used for preheating the *fuel*, consumes any carbonaceous matter that may be deposited therein. It may also be mentioned that the mechanical energy required to produce the necessary blast-pressure of the air and gas for this type of furnace and stove is compensated by the additional increase of temperature due to the combustion of fuel under a greater pressure.

LARGE FURNACES ON ALABAMA MATERIAL.

AS ILLUSTRATED IN PRACTICE BY ONE OF THE FURNACES OF THE TENNESSEE COAL, IRON AND RAILWAY CO., ENSLEY CITY, BIRMINGHAM, ALA.

BY FRED. W. GORDON, PHILADELPHIA, PA.

(Birmingham Meeting, May, 1888.)

THE heading of this paper was prompted by the knowledge that experience, up to this time, seemed to indicate that smaller furnaces were preferable for smelting the material of this section.

Since the reasonable success thus far achieved by the first furnace of this plant tends to change the above opinion, and unsettle the generally-accepted views of furnace-men as to "weak" coke, a few words as to the design of the plant and the mode of operation adopted may prove interesting.

The Coke.

As to the coke question, the Institute *Transactions* contain much interesting matter on the use and value of coke, which goes mainly to prove, and has largely contributed to establish the prevalent belief, that high crushing strength is an almost absolute requisite in coke. Southern furnace-men have had to face the battle with coke so weak that handling direct from the cars had to be adopted to save great waste; and yet they have had fair success. The knowledge of this work led the writer to fancy that there was, perhaps, too much stress laid on the requisite of strength, and that where strong coke was not obtainable, the other advantages of high, large furnaces, namely, economy of fuel and labor, were not to be abandoned, though it might be necessary to meet some troubles incident to the use of moderately soft fuel. Indeed, I am prepared to say, in the light of recent experience with this plant, that our cousins in the West of England, who raised their furnaces to eighty feet or so, and decapitated them afterwards, made an error in abandoning height. Their troubles must have arisen from the insufficiency of blast, interior lines not suitable for the high structure, or other causes or combination of causes, or from following out the practice suitable to the low furnaces, which may not have been proper for the high ones.

It may be admitted that the smaller the particles of coke, the greater will be the resistance to a given volume of blast; and that if the particles are very small, dirt-scaffolds are liable to be formed on the boshes. But with well-burned Pratt coke, furnaces of such height and diameter may be employed as will permit the attainment of all the economy of fuel hitherto reached in blast-furnace practice.

The analysis of the Pratt coke shows it equal to the Connellsville; and the burden carried by this furnace shows that the analysis is a correct indication of its calorific power. The only difference between the two cokes is, that a slightly less pressure, and consequent higher speed of driving might be attained with the former coke.

The Plant.

The plant at Ensley City consists of four furnaces, each 20 feet bosh and 80 feet high; sixteen Gordon-Whitwell-Cowper stoves, each 21 feet in diameter by 65 feet high; sixty-four return-flue boilers, 48 inches in diameter and 34 feet long (there being two flues, each 16 inches in diameter); twelve blast-engines, with 42-inch steam

cylinders, 84-inch blowing-cylinders, and 54-inch stroke. Other dimensions of furnaces are: Hearth-diameter, 10½ feet; charging-line diameter, 16 feet; bell-diameter, 12 feet; eight tuyeres, diameter 7 inches; tuyeres set 6 feet from bottom of hearth. Angle of bosh, 80°; it commences at the tuyere-line, and the in-wall is straight from top of bosh to charging-line.

These furnaces embody the patented and other features common to those built by Gordon, Strobel & Laureau, Ltd., of Philadelphia, the engineers of this plant.

The heating-stoves, four to each furnace (three being used at a time and found sufficient), are of the standard Gordon-Whitwell-Cowper type. There has been no modification of any kind in this type of stove since its introduction, three and one-half years ago; during which period seventy-six have been started or are under construction and near completion. A paper read at the Chattanooga meeting, in 1885, gives particulars of their construction.

Ores, Fuel and Flux.

The following are the analyses of the material:

Hard Ore from Red Mountain.

Peroxide of iron,	55.4481
Silica,	11.6
Alumina,	2.0019
Carbonate of lime,	29.2947
	98.3447

Metallic iron, 38.8 per cent.

Soft Ore from Red Mountain.

Peroxide of iron,	69.8278
Silica,	21.48
Alumina,	4.3321
Carbonate of lime,	.7875
	96.4274

Metallic iron, 48.88 per cent.

Limestone from Gate City, Ala.

Peroxide of iron,	.44
Silica,	2.79
Carbonate of lime,	94.22
Magnesia and alumina,	2.519
	99.969

COKE FROM PRATT MINES.

Moisture,	.175
Volatile matter,	1.18
Fixed carbon,	86.733
Sulphur,	1.081
Ash,	10.82
	99.989

The ash of this coke contained,

Silica,	5.455
Alumina,	2.957
Peroxide of iron,	1.193
Carbonate of lime,	.315
	9.920

The total ash being, as above, 10.82.

The sampling was done in the stock-house, very large samples being taken and carefully treated, to reach an honest average. Mr. Alfred F. Brainerd, chemist, of Birmingham, performed the sampling and analytical work.

BLOWING IN.

Thirteen cords of good wood was put in the furnace like a cribbing, but very loose; this reached up 20 feet from the hearth. On this was dropped 20,000 pounds of coke. Then followed 80,000 pounds of coke, mixed with 12 per cent. of limestone. Then,

	Coke.	Hard Ore.	Soft Ore.	Limestone.
10 Charges,	6000	3000	1500	1420
5 Charges,	6000	3300	1650	1490

The furnace was lighted at the iron-notch at 11.30 P.M., Thursday, April 5th, and at all the tuyeres at 3 A.M. on the 6th. At 5 A.M. filling was commenced again by putting in

	Coke.	Hard Ore.	Soft Ore.	Limestone.
5 Charges,	6000	3300	1650	1490

and the furnace was filled to within 12 feet of the top by 10 A.M. with

	Coke.	Hard Ore.	Soft Ore.	Limestone.
6 Charges,	6000	3630	1850	1560

During the combustion by draught, which was well-checked to give time for an adjustment of the connections, etc.,

	Coke.	Hard Ore.	Soft Ore.	Limestone.
4 Charges,	6000	3630	1850	1560

were made.

Blast was put on the furnace at 8 P.M., Saturday, April 7th, by running one engine, 25 revolutions, equal to 8500 cubic feet of air per minute by piston-displacement.

The temperature of the blast was not perceptibly above that of the atmosphere.

The first cast was 14 beds of white and mottled iron, made at 11.30 Sunday night, April 8th. At this time there were 40 revolutions, or 13,600 feet of blast, on the furnace; temperature, 700° F.; pressure, 2 pounds, the blast and its temperature having been increased simultaneously and as regularly as possible. Up to this time the furnace had taken:

	Coke.	Hard Ore.	Soft Ore.	Limestone.
10 Charges, ...	6000	3900	1950	1560
10 Charges, ...	6000	4200	2100	1560
10 Charges, ...	6000	4500	2250	1560
10 Charges, ...	6000	4800	2400	1560
10 Charges, ...	6000	5100	2550	1620
10 Charges, ...	6000	5400	2700	1680

A second cast, of 16 beds of No. 1 Mill, was made at 8 A.M. on the 9th, at which time the blast-volume was 18,000 cubic feet, the temperature 1000° F., and the pressure 4½ pounds; and the following charges had been filled in:

	Coke.	Hard Ore.	Soft Ore.	Limestone.
10 Charges, ...	6000	5700	2850	1740
10 Charges, ...	6000	6000	3000	1800
10 Charges, ...	6000	6300	3150	1860

Succeeding casts are shown in the following table:

No. of Cast.	Time.	Estimated No. of Tons.	Grade.	Remarks.
3	April 9, 3.30 P.M.	22	No. 1 Mill,	Up to the 8th cast the volume and temperature of blast had been maintained at 19,000 ft. and 1000° F. and the pressure had increased to 5 lbs.
4	April 9, 8.30 P.M.	24	Open Bright,	
5	April 10, 3.30 A.M.	24	Open Bright,	
6	April 10, 9.30 A.M.	24	Open Bright,	
7	April 10, 3.30 P.M.	23	Open Bright,	
8	April 10, 9.00 P.M.	23	Open Bright,	
9	April 11, 2.25 A.M.	23	Open Bright.	
10	April 11, 9.00 A.M.	24	Open Bright.	
11	April 11, 3.15 P.M.	28	Open Bright.	
12	April 11, 9.45 P.M.	24	No. 1 Foundry.	
13	April 12, 3.15 A.M.	24	No. 1 Foundry.	
14	April 12, 9.45 A.M.	23	Open Bright.	
15	April 12, 3.30 P.M.	26	No. 2 Foundry.	
16	April 12, 9.00 P.M.	26	No. 2 Foundry.	
17	April 13, 3.00 A.M.	25	No. 2 Foundry.	

The estimated amount of pig for each cast is not at all correct, and is below the facts, the scales for weighing the pig-iron from the casting-house not having been put in up to this time, and the company's record given above having been purposely placed below the actual.

During the time these casts were being made the furnace received the following charges:

	Coke.	Hard Ore.	Soft Ore.	Limestone.
10 Charges, ...	6000	6600	3300	1920
10 Charges, ...	6000	6900	3450	1980
10 Charges, ...	6000	7200	3600	2040
10 Charges, ...	6000	7500	3750	2100
10 Charges, ...	6000	7800	3900	2160
68 Charges, ...	6000	8100	4050	2220
68 Charges, ...	6000	8400	4200	2280
68 Charges, ...	6000	8700	4350	2450
68 Charges, ...	6000	9000	4500	2520

After several days working of this last charge it was decided that it was too heavy to be used in making foundry iron, and the charge of 8700 hard ore, 4350 soft ore, and 2450 stone was returned to, and continued up to the present writing (May 2d). This the furnace has been found able to carry in good style.

Yesterday (May 1st) the furnace was under the last-named burden; the volume of blast was 22,000 cubic feet, the temperature 1400° F., and the pressure 8 to 9 pounds. This is kept as constant as possible, neither the burden, temperature nor volume of blast having been changed for a week past.

The make of the furnace, 165 tons yesterday, is nearly an average for each day's work.

The composition of the slag, intended to be about 40 silica, has changed but little. We started with it a little too high in silica, but have now brought it down to the required condition.

I may say here that the variation in the composition of slags in the furnaces in the Birmingham district is exceedingly slight, the percentage of the silica rarely varying more than 2 per cent. This I particularly wish noted, as it is the surest proof that the ores of this district are very regular, in spite of the accepted opinion throughout the North that they are extremely irregular, and that this irregularity has caused much of the trouble heretofore experienced in smelting them.

At our Chattanooga meeting in 1885, the President, Mr. Bayles, estimated the fuel-consumption of Southern furnaces to be from 1¾ to 2 tons of coke per ton of pig-metal. This statement was correct. The

working of this furnace, however, proves the possibility, at least, of a pound of iron from a pound of coke; for the above mixture is yielding 46 per cent. of pig-iron, and figures 6003 pounds of iron to 6000 pounds of coke.

Discussion.

Alfred F. Brainerd, Birmingham, Ala.: A number of attempts have been made in the Birmingham and other districts to improve the quality of the coke by washing the coal. The Woodward Iron Company is regularly washing its coal, with decidedly beneficial results. The Watts Coal and Iron Company has made an experimental success in washing its coal, and will put in washing machinery. The Cahaba Coal and Railroad Company has successfully washed its coal, but owing to the increased cost has decided to abandon the project and to improve the quality of its coal by assorting it, picking out the slate, etc. The Milner Coal and Railroad Company has made a partial success in washing, but has temporarily abandoned its experiments in this direction, intending to take the matter in hand again in the future. The Coalburg Coal and Coke Company made, in 1885, many ineffectual and unsuccessful attempts to wash its coal, and abandoned the process.

In connection with the analyses of stock in Mr. Gordon's paper, it should be observed that the ore was wet from recent rains, and the damp samples were analyzed without drying; hence the soft ore, particularly, shows rather less iron than was expected of the average. But the analysis shows the correct average of the stock-pile as it was actually charged into the furnace.

George Jamme, Dayton, Tenn. (revised, July, 1888): It is now about one month since the Birmingham meeting of the American Institute of Mining Engineers took place, and the reported success of the Ensley furnaces continues. I see nothing, however, to change materially the opinion expressed by me at the meeting on Mr. Gordon's paper, which appeared to me at that time to be rather a "headquarters-in-the-saddle" sort of bulletin than a convincing and established proof that high and large furnaces are best for Alabama materials.

I have too much respect for Mr. Gordon's evident and remarkable abilities, and courage in expressing his conviction, to criticize captiously his expressed opinions, even had any reverse attended the operations of the two Ensley furnaces now in blast. Any furnace is subject to accident, and in such matters conclusions should not be

hasty; therefore, I propose to give my views without reference to the events of the past month's career of the Ensley furnaces, simply on general principles, as I gave them at Birmingham.

Mr. Gordon, in introducing our English cousins, and deprecating their action in decapitating their high furnaces, permits a wider range of criticism, and gives an opportunity to correct a prevalent notion that the ores used in the furnaces of the Birmingham district and the Chattanooga district of the South, are one and the same in composition, and that the fuels used in these districts are also the same. I do not know that Mr. Gordon holds this opinion, but I do know that it is entertained by our cousins abroad and by many of our Northern friends.

Coke.—Judging from what we have seen in the way of coke at the Ensley and Pioneer furnaces, the coke used in Alabama is not a soft coke, or, if it is, it need not necessarily be so. There are many places on this continent where coke much softer and much more poorly put together is used, and still money is made in the manufacture of pig-iron. If the coke at the above-mentioned furnaces is structurally soft, it is due to bad treatment of the coal, either in mining or immediately afterwards. It is due to the presence of slate particles, or rather pieces, more or less large, that place themselves across the columnar structure, arrest the action of the incandescent gases in their work, and by their resistance to parting in the columnar direction cause the coke to break across when being pulled and before the coke leaves the oven. The result is a general friability caused by the presence of these slate particles or pieces. This, however, can be remedied, so that Pratt coke need not necessarily be called a soft coke; and there is no reason why it cannot be made equal to fair competition with Connellsville coke. It is as low in ash, about as high in carbon, it has the proper ring, is equal to Connellsville in cell-capacity, and I believe as able to resist crushing-test as any Connellsville coke. Washing cannot be used to make it any better, for water is not abundant enough in the South near the Pratt mines to permit that process, and if there is anything in the claim that the Connellsville ash "makes" the Connellsville coke, it may as well be applied to Pratt coke.

In the Chattanooga district the condition of the coke is very different. Ash is more abundant. It often rises to 23 per cent. when the miners are careless, and 19 per cent. may be called normal. In this district high and large furnaces have been advocated, but have not been as successful as was expected, not altogether because of the

condition of the coke, but for reasons relating to the ores of the district. The use in them of ores of the Alabama district has not, however, altered the case. Coke made under the same conditions of manufacture as those of the present practice in Alabama and Tennessee, would work exactly the same if made of coal from other districts, and with such coke, the dust-catchers, absolutely necessary in Alabama as well as Tennessee, would be required in districts where they are not yet known, to relieve the gorged stomachs of furnaces. With this evil furnace-managers in Southern practice are well acquainted. Without the convenient dust-catchers the coke would fill the flues with a dust which, moreover, is not found in the South to be as good a fertilizer as the dust found in the flues of Northern or Eastern furnaces.

Plant.—No impartial visitor could find serious fault with the plant at Ensley, or, in fact, with any of the recently-built furnaces in the Birmingham district. When the Ensley plant is complete, it will be, without doubt, the finest in the world. Nothing has been spared to endow it with full power in every requisite. The plant is simply "monumental." Is this, however, a criterion of economy? Is it a commercial perfection? I think not. I take it for granted that furnaces are not built, like monuments, to be gazed at and admired. They must have a successful industrial life apart from natural advantages of location; in other words, they must be built to make money, and as much of it as possible—not in a spasmodic manner, or in a steeple-chase style, resulting in such wrecks as are to be seen in and around Birmingham, in furnaces that are hardly a decade old, and are, by the way, not now ostentatiously exhibited by our good friends in that city. It is quite certain that the large furnaces that have been in operation in Birmingham for several years have not been as successful as the smaller ones, either in freedom from disaster, regular running, or a good quality of metal. With smaller furnaces, and more of them, the risks would be lessened; depressions would be tided over more easily by a more elastic control of production, and the first cost of erection would not be enhanced in the same degree as the risk and loss of time in recovering from "trouble," which are disproportionately large with furnaces of excessive width of bosh or great height. For instance: Was the Ferry Hill furnace, in England, with 103 feet height and 27 feet of bosh, a success, as compared with the 80 feet height and 18 and 20 feet bosh of Middlesborough, both sizes working on similar materials? Height in a furnace, as I think Mr. Gordon will agree, is correlative to the ease

of reducibility of the ore in use. Will the large Ensley be as successful in the use of the red fossil ores as the Belgian furnaces are, which use the Minette, and make respectable quantities at a price that even the most enthusiastic real-estate agent in Birmingham or Sheffield has not yet dared to utter? The Belgian furnaces are not high furnaces; their height reaches only a few feet above the top of the boshes of the Ensley; their stove-capacity is very modest; but their low consumption of fuel is remarkable, and may yet be envied by our Ensley friends when they look behind the returns.

In the manufacture of foundry-pig (and that is the kind of stuff the Ensley furnaces will have to make) rapid driving becomes an absolute necessity, a *sine qua non* of good foundry-iron. Anything that retards the pace gives time for saturation of the metallic iron with silicon, and is the cause of production of pig-metal with high silicon. Mr. Gordon couples large diameter with height; these assist each other in preparing disasters; rapid driving is accompanied with abundant blast, the section of plastic material is carried higher, and more readily within the reach of increased burden and cooling influences, and less under control than in the smaller furnaces.

Mr. Gordon does not seem to believe in the irregularity of composition of the Southern ores. With more extended actual practice he would have found that they do vary greatly (especially the hard ores) in the amount of lime; and, unfortunately, the difference does not carry with it a corresponding change in the proportion of silica. It is in this respect that I fear the practice of carrying a high percentage of silica in his calculated burden for slag will bring him to grief, or at least those who under his inspiration are in pursuit of large production. I have not had opportunity to obtain an analysis of the pig-iron we saw at Ensley, nor has the verdict of the captious customer had time to become known. Even at this time of writing I cannot see yet that the success so far obtained at Ensley is due to the proportions of the furnaces. The South Chicago furnaces, which have been proposed to us as examples of successful running, are not high furnaces in the proportions of the Ensley; and I would ask Mr. Gordon whether the brilliant success at the Chicago furnaces is not due to a proper mode of distribution of stock at the top, rather than to the low percentage of lime, and the special conditions and relations of iron and cinder produced. This mention of the South Chicago furnaces may seem digressive in discussing furnaces working on Alabama materials; yet it is a fair parallel argument. Be-

fore ranging myself to Mr. Gordon's opinion that high and large furnaces are best for Alabama materials, I shall require further evidence produced by experience. But I feel quite sure that no one will yield with more grace than I, if any good way can be established to relieve the Southern furnace-managers of some of the ills which, in the words of our former President, Mr. Bayles, make their hair turn prematurely gray.

F. FIRMSTONE, Easton, Pa.: Coal-washing in this country has not, in very many cases, proved successful.

No doubt, this is in part due to the machinery having often got into the hands of managers who did not understand the principles of mechanical separation; but I believe, in many instances, machinery has been put up in places where a careful preliminary examination would have made it clear that a good result was, in the very nature of the case, quite out of the question.

Four years ago we tested a coal-seam in West Virginia, which was very favorably situated for working, and of which we owned a large area.

The coal coked well, but the coke contained nearly 16 per cent. of ash, and showed so many pieces of slate that the chance of reducing the ash, by washing, to 6 or 7 per cent. (which was the content of the coke we were making from another seam) seemed good.

As a preliminary step, a large sample was treated with chloride of calcium solution in Dr. Drown's laboratory,* which gives, of course, a far better separation than the most perfect jigging; but it was seen at once, that to bring the coke up to the desired standard there would be a loss of from 21 to 28 per cent. of the coal, which was, under the circumstances, not to be thought of.

Had we fallen into the hands of an expert, we would, in all probability, have arrived at the same result by running a dressing-plant.

It seems to me, that those who contemplate coal-washing ought, by all means, to submit the coal to this simple preliminary test, whereby they may know beforehand that the washing can, at the best, do for them not quite so well as the separation by dense solution.

T. H. ALDRICH, Cincinnati, Ohio: In this discussion the fact seems to be lost sight of that the purity of the coke depends upon careful mining of the coal. It is my belief that the Pratt and other seams would produce a first-class coke if the coal was properly mined. The analyses of Alabama coals, in general, show that a normal coke

* See *Transactions*, vol. xiii. (1884), p. 341 *et seq.*

would have less than 10 per cent. of ash. Careless methods, and lack of competition, are more responsible for dirty coke than the coals themselves.

E. C. PECHIN, Cleveland, Ohio: I am compelled to take issue with Mr. Gordon on his statement that the "Institute *Transactions* contain much matter on the use and value of coke, which goes mainly to prove and has largely contributed to establish the prevalent belief that high crushing-strength is an almost absolute requisite in coke."

So far as I am familiar with the statements of the *Transactions*, they have mainly led to opposite conclusions.

As far back as 1873, Mr. T. F. Witherbee gave an interesting account of the substitution of anthracite, on top of charcoal, in the Fletcherville furnace, Oct. 6–11 (*Trans.*, ii., 71).

"The anthracite was charged directly on top of the descending charges of charcoal, and notwithstanding the immense weight of material as compared with the weight of charcoal charge, no evidence of crushing was apparent, and the furnace, to say the least, never worked any better." One-half of the charcoal used was from soft woods, pine, hemlock, and spruce.

At the May meeting of 1878, Professor J. A. Church presented a carefully prepared paper on "The Mode of Combustion in the Blast-furnace Hearth" (*Trans.*, vii., 33). In it he argues for a light, porous, delicate, and easily ignited coke, so that it might be more readily acted upon by the oxygen of the blast. With this, at present, we have nothing to do. The following is, however, germane: "The reason why a similar course has not been followed in coke-practice, is a mistaken notion that such fuel is not strong enough to carry the weight of the ore. But this chimera of weak fuel should be passed by as belonging to the traditional period of furnace-management. . . . . This mistaken idea has maintained itself with singular perversity."

Mr. F. P. Dewey, in his paper on "Porosity and Specific Gravity of Coke," June, 1883 (*Trans.*, xii., 111), only refers to the crushing properties of coke as follows: "The limit (amount of cell-space) is reached when the cell-walls become so thin that the coke is unable to bear the burden in the furnace."

Mr. John Fulton, to whom we all are so much indebted for his painstaking and valuable researches, in October, 1883, contributed an important paper on "The Physical Properties of Coke as a Fuel for Blast-furnace Use" (*Trans.*, xii., 212).

The only allusion to a crushing effect is in the following: "Cubes of coke were used for the reason that they afforded the best form

for making the additional test of *compressive strength*, indicating their relative hardness, and consequent ability to resist the action of the gases in the upper regions of the furnace, and sustain the burden without crumbling."

He found the height of furnace-charges supported without crushing for Connellsville coke 114 feet, West Virginia 113 feet, and in the Alabama coke of that day, 87 feet, thus practically eliminating, even if there were anything in it, the crushing question from any furnace now in existence in the United States.

It would therefore appear that so far from differing, the Institute and Mr. Gordon are quite in accord.

As to the height of a furnace, it would seem to me that this should properly be regulated by the character of the ores to be smelted in it.

It has been clearly established that hematites reduce with great rapidity, and do not require to be kept in contact with the reducing gas anything like as long as specular, carbonate, or magnetic ores. If, with given materials in a properly constructed furnace, the gases go off at a temperature incapable of any further reduction, and of the proper composition, at 70 feet, there is no use in building to 80 feet.

The red ores in the South would seem to stand in the list of easily reducible ores, and the inference would be that, with a properly-made coke, and with similar heat, a furnace 70 feet high ought to turn out relatively more iron per cubic capacity than one of 80 feet. Certainly, such a furnace would be less expensive and more readily handled in case of trouble.

The only way to settle the question would be to have two furnaces, one of 70 feet and one of 80 feet, worked under the same conditions and the results noted.

As you have very kindly allowed, Mr. President, considerable latitude in these discussions, will you permit me to allude very briefly to a question which has received considerable private consideration among the members present, viz., the effect of poorly-made coke in furnace-use?

There seem to be two very serious objections to the use of soft coke.

Bell has very clearly shown the solvent effect of carbonic acid gas upon soft coke in the upper part of the furnace, and that there follows a positive and substantial loss of fuel. As Fulton well puts it: "Every pound of coke consumed by the gas in the upper region of the furnace is *a double loss*, by the reduction of the temperature

where the action takes place, and the loss of fuel, which should have been burned near the tuyeres" (*Trans.*, xii., 217).

Secondly, a highly objectionable mechanical action takes place. According to Bell, if the ore has been properly reduced, at 25 to 30 feet from the top of a 75- or 80-foot furnace it is in the shape of a powder. Under this zone the decomposition of the limestone takes place, resulting in lime, a brittle substance that in its descent must rapidly pulverize. The coke is the only material that keeps these powders from aggregating, and affords interstices for the passage of the blast. If soft coke is used, it breaks up in the fall, and the soft edges are worn off in its rapid descent; these help to fill up the interstices, growing smaller and smaller, practically packing the mass, and requiring stronger blast, and oftentimes causing grave irregularities in working, if not scaffolds.

In using soft coke, the higher the furnace the greater the difficulty; not because it crushes, but because there is so much greater space for pulverizing, running and packing; soft coke wastes in every direction, from the time it leaves the oven until what it has made comes out of the furnace for grading and sale. A thoroughly good coke ought to be made out of the Pratt seam of the Warrior coal-field; and if it is not, the users of it cannot be pitied for the losses sustained.

R. W. RAYMOND, New York City: I entirely agree with Mr. Pechin as to the baselessness of the notion that higher furnaces mean a more crushing weight on the burden, and, for that reason, require special strength of the coke. The longer travel through a high stack may, indeed, through increased abrasion, pulverize the fuel to a greater extent; but there is no extra weight—the weight due to extra height being wholly carried by the side walls. This principle is a familiar one to civil engineers (see *Transactions*, xiii., 493). It is the reason why buildings overloaded with loose grain burst their walls, not their floors. It is impossible to bring upon a floor, enclosed with side-walls, a greater weight of such piled loose material than the weight of a pile the bottom of which just covers the floor. The truth of this can be easily shown by a simple experiment: Take a glass tube (say a lamp-chimney), clamp it in a vertical position, and adjust at the bottom the scale of a balance so that when sand is poured into the tube till, taking its natural *talus*, it just covers the bottom (that is, the part of the scale within the circular bottom-edge), the counterweight holds the scale against that edge. This having been secured, the tube may be filled with sand to the very top without producing any movement of the scale. The tube

takes all the extra weight, which, being thus independently supported, does not affect the bottom.

Any one who has looked into the hearth of a blast-furnace in operation will admit that the conditions as to downward pressure are similar. The stock arches itself in the furnace; and though this is doubtless less uniformly done than in the case of sand in a tube, because the size of the pieces is less regular (and for other reasons), I think the difference in the two cases strengthens the force of the analogy; or, in other words, the downward pressure in the furnace is proportionally less than in the experiment I have described.

MR. GORDON (Communication to the Secretary, September, 1888): Every one at all familiar with the working of iron-ores of various compositions, etc., must admit that there is a vast difference in their reducibility. Mr. Jamme and Mr. Pechin seem to think I have either not recognized this fact or have not given it full weight. At least their criticisms involve the inference that as good economy could be obtained on like materials with furnaces of less height. The height of 80 feet was adopted at Ensley, that good fuel-economy might be secured; and the result shows we were not mistaken. I am persuaded that the red fossil-ores of Alabama are readily reduced, and that better results may be obtained from them with low furnaces than has hitherto been the case; but, in view of the comparative results so far, I do not think that the Ensley furnaces are too high.

What has cubical capacity to do with furnace-production? It may be something to boast of, that great products are made from small furnaces; but for a given product on a certain fuel-consumption experience has taught us that it is desirable in every way to have a certain engine-power, boiler-power, hoisting-capacity, blast-heating capacity, and size of cast-house. It is therefore quite evident that the plant cannot be materially reduced in cost by simply reducing the furnace-stack. Mr. Pechin suggests 70 feet instead of 80 as the height, since it would cost less and perhaps work better. The saving on the Ensley plant by this change would not exceed two per cent. on the whole.

Mr. Jamme cites the Ferry Hill furnace as too large, compared with the moderate-sized furnaces at Middlesborough. But the latter are almost identical in size with those at Ensley. This reference is therefore not well taken. And I am not sure, in the light of the great increase of size from the 10-ton charcoal-furnaces of a few years ago to those at Ensley, and still larger ones, that if the Ferry Hill had been similarly proportioned and blown as rapidly per

unit of cubical capacity as those at Middlesborough, it would not have done, at least, quite as well.

I believed the Pratt coke and the red fossil-ore of Birmingham would work well and economically in large furnaces, and so they did. No amount of argument is equal to practical results; and until the small furnaces produce pig-iron with as low a consumption of fuel, labor and interest on plant as those at Ensley, I cannot admit that I erred in recommending 20 x 80 feet furnaces for these materials. There are plenty of furnaces working similar material varying from 55 to 75 feet in height, from which Mr. Pechin can get the practical data he suggests for a determination of the question. These have been operated for a number of years, and, so far as I can learn, by no chance has any one of them got the fuel-consumption as low as has been reached at Ensley.

Mr. Jamme very unkindly calls the plant "monumental." It was not so intended, either by the designers or the Company. Modern American blast-furnace practice calls for all there is, or is intended to be, at this plant, to equip four such furnaces. Three 84-inch engines; four 21 x 65 feet fire-brick stoves; sixteen 48 x 34 feet double-flue boilers, with moderate-sized cast-houses, and wooden stack-house (really too small), cannot be considered too much for the requirements. The plant is roomy, but this room adds nothing to the cost, except what is due to longer cold-blast mains, flues and water pipes—a very small sum indeed. To novices this plant may seem immense; but engineers should take it in detail, if any criticism is to be made. It has nothing excessive, except perhaps the fourth stove to each furnace; and this cannot be deemed an extravagance, seeing that so many experienced manufacturers consider this extra stove advisable. The Edgar Thomson Steel Co., the Pennsylvania Steel Co., the Cleveland Rolling Mill Co., and the Pioneer Iron Co., are examples.

If "rapid driving is a necessity for foundry-pig," how would Mr. Jamme do it with less support than there is at this plant?

I certainly couple height with large furnaces, not "large diameter with height," as Mr. Jamme says; but I do not admit that these assist in preparing disasters.

Large furnaces were condemned wholesale some years ago. When the Isabella and Lucy furnaces at Pittsburgh were built, being 20 x 75 feet, their projectors were condemned without stint; and the trouble experienced in running them was largely attributed to their great size. Yet to-day this is a favorite size, and furnaces of these and larger dimensions show the best of records.

Mr. Jamme says, "Anything that retards the pace gives time for the saturation of the metallic iron with silicon, and is the cause of the production of pig-metal with high silicon." I have always understood that if silica is reduced the silicon would impregnate the iron almost instantaneously, and that high speed of driving may have nothing to do with the proportion of silicon in the iron made, or rather that extra-slow driving, cooling the hearth, will reduce the proportion of silicon, if all other conditions remain constant. Moreover, in fast or slow driving, the percentage of silicon is practically under control by regulating the composition of the slag and the temperature of the hearth, the latter being approximated to the point desired by adjusting burden, blast-heat and speed of driving. Either excess or lack of blast will tend to decrease the hearth-temperature.

NOTES ON THE IRON-ORES, FUELS, AND IMPROVED BLAST-FURNACE PRACTICE OF THE BIRMINGHAM DISTRICT.

BY ALFRED F. BRAINERD, BIRMINGHAM, ALA.

(Birmingham Meeting, May, 1888.)

THE subject of the supply and the quality of the iron-ore and coke of this State has suffered exaggeration and misrepresentation in both directions. Unsophisticated persons have made extraordinary reports of quality and richness, which could not be supported by facts; while, on the other hand, experts have selected inferior samples, and upon the average of these, which was not by any means the average known to our practice, have based comparisons favorable to other States. Our furnace-records give much better testimony as to the run of our ores, the calorific duty of our coke, and the burden and blast which it will bear. Some of them show poorer ore-mixtures than others; but it is safe to say that the Birmingham furnaces produce from 40 to 50 per cent. of iron from a mixture of equal parts of the leanest and the richest ores, as classified in large quantities. And certain performances on record here, while they do not represent the regular or average practice, may be fairly urged as proof of excellence of raw material, since they would otherwise have been impossible, even as exceptional achievements. Thus one of our furnaces has made iron with 2500 pounds of coke to the ton of 2300 pounds (Mr. Stratton, 1888); another, with a ton of coke to the ton of iron (Mr. Gordon, 1888); another, with 1.27 ton of coke to the ton (2300 pounds) of iron, the product in the last case being from

a 11 by 55 ft. furnace, 66 tons for 24 hours, and the blast-temperature 1150° F. (Mr. Edwards, 1888). Such low fuel-consumption is certainly not universal, and probably not advisable. Some of our best managers prefer to use more fuel, and secure a higher (grayer) average pig, with greater "margin" in case of furnace troubles. But, I repeat, the above figures could not be realized with bad stock.

In explanation of the analyses given below, I recall briefly the geological character and position of our ore-deposits. These are distributed in ridges and belts, running northeasterly and southwesterly —a distribution which naturally follows from the geological structure of the State, the formations of which have that general course.

The Red Mountain near Birmingham is the most important outcropping range of the Clinton fossil-ore, which here attains exceptional thickness and value, and furnishes the principal supply to the Birmingham furnaces. The ore-bed in this mountain is from 10 to 20 feet thick, and the depth to which it has been mined does not exceed, say, 300 feet.

For the first 100 feet from the surface it carries "soft" ore, containing little or no lime carbonate, and averaging from 50 to 55 per cent. in iron. From 100 to 200 feet, the lime carbonate increases rapidly, until the bed carries about 40 to 45 per cent. of it, and only about 25 to 35 per cent. of iron. This is called "hard" ore. In furnace-practice it is not unusual to mix the hard and soft ore in equal parts, utilizing the lime of the hard ore as flux. One furnace here used three parts hard ore and one soft ore, then three hard ore to one brown ore, alternating (Mr. McCabe, 1885). Sometimes as little as 250 pounds of limestone is added to 4600 pounds of such an ore-mixture. But mixtures and burdens vary greatly in the district, nearly every furnace-manager having his own ideas on the subject.

Most of the coke is now made from the Pratt seam, worked at the Pratt mines by slopes and shafts. The seam is 4 to 4½ feet thick, and the deepest shaft reaches at about 350 feet the bottom of the basin, which is seven miles wide. The daily product is 3000 to 3500 tons of coal, the coke from which supplies the Sloss, Alice, Ensley, Mary Pratt, and Eureka furnaces.

In the Blue Creek basin, which is three miles wide, the seam is nine feet thick. It has been recently developed to supply the De Bardeleben Coal and Iron Co. for its two blast-furnaces, rolling-mills, etc., at Bessemer. The bottom of the basin is 1105 feet from the surface on the incline (varying up to 15°), or about 250 feet vertically.

The Coalburg mines are located on a part of the Pratt seam, which

shows, however, at this point a different physical and chemical character.

The Newcastle and Black Creek seams are both mined by the Milner Coal, Iron and Railroad Co., at Newcastle, Ala. One seam is six feet, the other four feet thick. Col. John T. Milner has produced from both washed and unwashed coal a good coke, which, on a 48-hour trial in the Mary Pratt furnace, worked, under similar conditions, as well as the Pratt coke, carrying about the same burden, performing the same calorific duty, and producing an excellent grade of iron.

The Woodward Iron Co. cokes the coal of its own mines.

The following analyses of flux, ore, and fuel, made by me, are offered as fair samples of general quality:

I. Limestone.

Both of these analyses represent the limestone used by the Tennessee C. and I. Co., No. II. being from a sampling of the stock-pile.

	I.	II.
FeO,	0.440	0.703
SiO_2,	2.790	3.380
Al_2O_3,		1.297
CaO,CO_2,	94.220	92.482
MgO,CO_2,	2.519	0.757
Undetermined and loss,	0.031	1.381
	100.000	100.000

II. Iron-Ore.

I. Soft ore, from the stock-pile of the Tennessee C. and I. Co.
II. Hard ore, from the stock-pile of the Tennessee C. and I. Co.
III. Red ore from between Lakeview and Avondale (owned by Pioneer I. and Manuf. Co.).
IV. Red hematite from a vein eight feet thick in Eastern Alabama.

	I.	II.	III.	IV.
Fe_2O_3,	69.828	55.448	75.543	90.778
SiO_2,	21.480	11.600	18.000	5.060
Al_2O_3,	4.332	2.092	Undetermined.	Undetermined.
CaO,	0.788	29.295	"	"
Metallic iron,	48.879	38.786	52.880	63.545
Phosphorus,			0.130	0.056

Analysis IV. represents a range in this State, which I believe to contain Bessemer ores, and I deem it not premature to express such a judgment upon the results of several analyses. Railroads now projected, or in progress of construction, will open a number of parallel ranges of both brown and red ores, among which it is confidently expected that some will prove low enough in phosphorus for Bessemer and open-hearth use.

III. COKE.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.	X.
Moisture	0.165	0.170	0.500	1.60	0.075	0.400	?	0.250	0.200	0.150
Volatile	1.190	1.179	1.700		2.720	1.800	3.600	0.400	1.600	1.120
Fixed Carbon	86.744	89.133	90.340	83.43	91.355	89.533	78.973	92 327	92.304	91.469
Sulphur	1.081	0.858	0.800	0.72	0.700	0.867	1.947	0.823	1.116	0.661
Ash	10.820	8.660	6.660	14.25	5.150	7.400	15.480	6.200	4.780	6.600
	100.000	100.000	100.000	100.00	100.000	100.000	100.000	100.000	100.000	100.000
Analysis of Ash.										
SiO_2	5.455	3.370	Phospho-					2.980		
Al_2O_3	2.957	3.510	rus in III.					1.302		
Fe_2O_3	1.943	1.152						1.412		
CaO	0.315	Undetermined	0.050					0.420		
MgO	0.150							0.086		
P_2O_5		.073		.1145						
	10.820	8.105						6.200		
Color	Ginger.	Ginger.	Ginger.	Red.	Brown.	Brown.	Slate.	Ginger.	Gray.	Drab.

I. Pratt coke sampled from the stock-pile of the Tenn. C. and I. Co.
II. " " " " " " Sloss Furnace Co.
III. " " " " coke-ovens " " " " "
IV. " " " " stock-pile " Alice furnace (analyses, 1885).
V. Cahaba coke, analysis of 1887.
VI. " " " 1888.
VII. Coalburg coke,
VIII. Blue Creek "
IX. Watts " (from washed coal).
X. Newcastle " " " " analysis of 1887.

With improved labor and judicious mining and sorting, the Alabama coke could be made to show from 8 to 10 per cent. of ash in average samples at the furnace, instead of 10 to 12, as is probably now the case. A reduction of 3 per cent. in ash has indeed been accomplished during the past three years for the coke from the Pratt mines, by improvement in labor and extra precautions in mining and sorting coal. The furnaces feel the difference in the smaller amount of coke required for the same duty. A Birmingham foundry-manager recently remarked that several years ago he succeeded in melting in the cupola only four pounds of iron with one of coke, whereas he can now melt eight pounds of iron with one pound of coke. This compares well with Northern foundry-practice.

Our mining labor is undoubtedly inferior, in many places, at least, to the free, white, skilled labor, especially the Cornish and Welsh labor, trained through generations, which is available at the North. Convict labor of whatever race is certainly inferior; and our free labor, whether white or colored, is, in general, relatively inexperienced. Yet, with these drawbacks, our mining is cheaply done;—I think more cheaply than elsewhere, if such favored localities as Lebanon, Pa., be excepted. And I have not heard that the ability to mine iron-ore at the Cornwall banks of Lebanon for twenty-five cents or less per ton has greatly benefited the purchasers of the ore, though it has, of course, brought wealth to the owners of the mine and of the furnace receiving ore from it at cost.

Having been brought up in the North, I took pains, four years ago, to investigate the prejudice which existed against Southern irons. After many inquiries among large consumers, I was satisfied both of the fact and of the absence of a solid foundation for it. Indeed, the increased market for such irons in all accessible parts of the North shows that when they can be bought at advantageous prices, the prejudice vanishes.

NOTES ON THE GEOLOGY AND ON SOME OF THE MINES OF ASPEN MOUNTAIN, PITKIN CO., COLO.

BY CARL HENRICH, M.E., NOBLE, ILL.

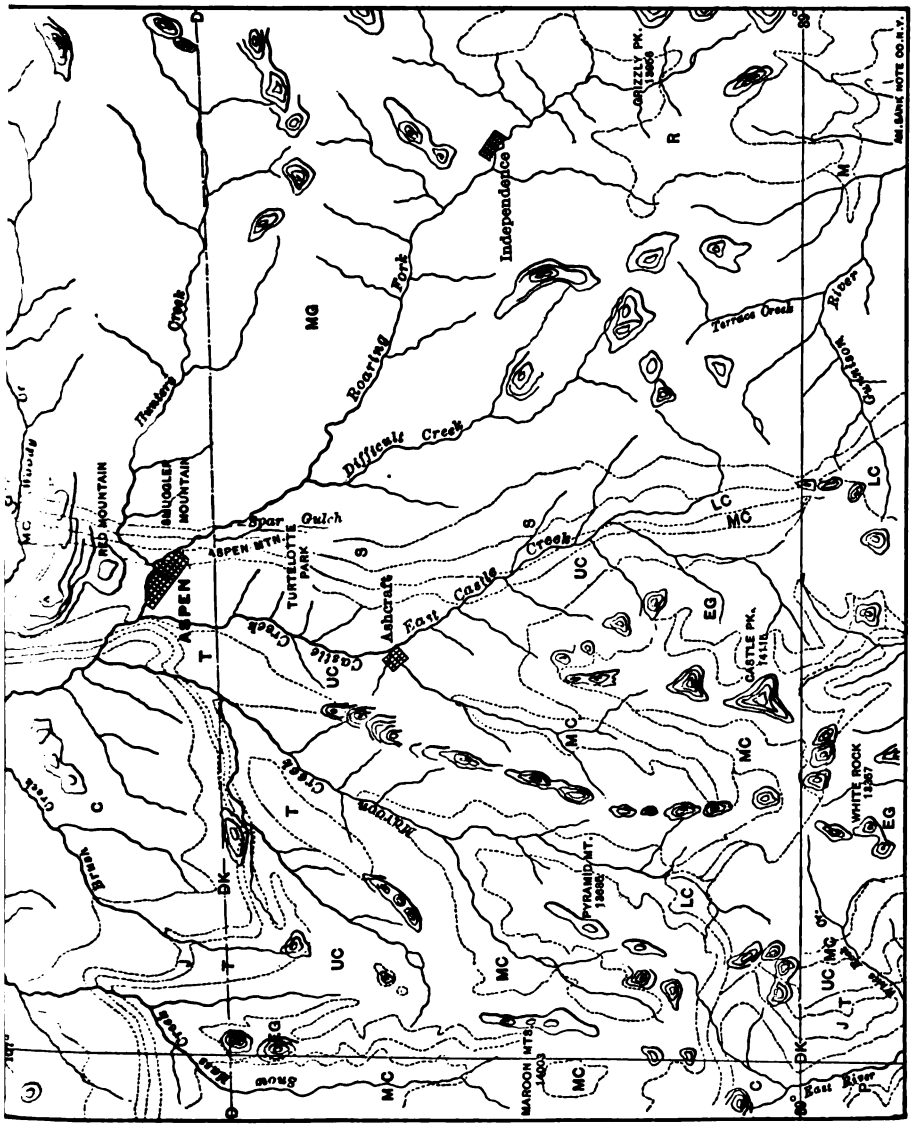
(Birmingham Meeting, May, 1888.)

I. INTRODUCTION.

ASPEN, the flourishing mountain- and mining-town of Pitkin county, Colorado, is located in the valley of the Roaring Fork, 41 miles above Glenwood springs, where that stream empties its waters into Grand river, a tributary of the Colorado. The Roaring Fork heads on the western slope of the Sawatch range near Grizzly peak, and runs in a general northwesterly direction to its junction with the Grand river. Of its numerous tributaries, Castle creek joins it just below the town of Aspen, the town being built on an extensive flat bounded on the northeast by the Roaring Fork and on the west by Castle creek. The latter heads at the junction of the Sawatch and the Elk mountains and flows northward between the two chains, until it empties into Roaring Fork. Opposite the town of Aspen, Hunter creek, coming westward from the western slope of Mount Massive, joins the Roaring Fork. About a mile north of Aspen, the bed of Hunter creek is at least 300 feet higher than that of the Roaring Fork at Aspen. Hunter creek forms thus, at its mouth, one of the best water-powers in Colorado. A portion of it is now utilized for electric lighting in Aspen, while in the near future, the whole of this magnificent natural power will be applied, through electric transmission, for hoisting, pumping and ventilating at different mines of the district.

The mountain opposite the town of Aspen, on the east side of Hunter creek (which near its mouth flows southerly), is called Smuggler mountain, from the famous mine at the foot of it. On the other, or western, side of Hunter's creek (see map, Fig. 1), a higher mountain rises, on the slope of which the regular dip of the Triassic or Upper Carboniferous red sandstone strata, alternating with white layers of the same rock, can be clearly seen from town. This mountain is called Red mountain, from the prevailing color of its rocky

sides. South of Aspen, Spar gulch, a dry ravine nearly parallel with Castle creek, separates Aspen mountain from the eastern part of the mountain spur, dividing Castle creek from Roaring Fork and



Sketch-Map Indicating the Geology of Aspen and Vicinity. (From Hayden's Geological Survey.)
S, Silurian, including Cambrian; LC, MC, and UC, Lower, Middle, and Upper Carboniferous; T. Triassic; J, Jurassic; C. Cretaceous; MG and EG, Metamorphic and Eruptive (diorite or porphyry) Granite; R, Rhyolite.

Difficult creek, another tributary of the latter. (See Geological Map, Fig. 1.) All that part of this ridge lying west of Spar gulch and

Turtelotte park (at the head of Spar gulch) is called Aspen mountain. This is again divided by Pioneer gulch (a steep depression, running from the town up the north slope of Aspen mountain to its crest, and parallel with Spar gulch), into Aspen mountain proper, between Spar and Pioneer gulches, and West Aspen mountain, west of Pioneer gulch.

The summit of the ridge of Aspen mountain runs nearly north, gradually descending, while the crest of West Aspen mountain, forming a narrow ridge, and in places a sharp escarpment, turns off from Aspen mountain to the northwest, descending rapidly in its course, with steep slopes and precipitious bluffs on both sides, and finally ends abruptly about half a mile above the junction of Castle creek and Roaring Fork, close to Castle creek.

The Aspen ore-deposits were discovered in 1879 by prospectors whom the Leadville excitement, then at its height, had stimulated to search for "carbonates" and "contacts," across the Sawatch range and along the belt of Silurian and Carboniferous rocks, depicted on the maps of Hayden's Geological Survey. (See Hayden's map and section of the Aspen region, Figs. 1 and 2.)

The town of Aspen was founded in the same year, and the struggle for prosperity began. The greatest drawback of the camp was its difficulty of access. Before the advent of railroads, two routes were open for travel into or out of Aspen. The original settlers of Aspen came in over two different trails, both afterwards converted into wagon-roads. The nearest route from Leadville was by way of the Twin Lakes, thence following up Lake creek and its northern fork to the foot of Hunter's pass, thence across the pass (11,700 feet high) to Independence, a then flourishing mining camp with gold-quartz veins, and down the valley of the Roaring Fork to Aspen. The distance from Leadville by this route is in the neighborhood of 50 miles. After the completion of the Denver and Rio Grande railway to Leadville, Granite, 15 miles below Leadville in the Arkansas valley, became the nearest railroad point, the distance over the above route, via Hunter's pass, from Aspen to Granite being estimated at 30 to 35 miles. The stage used to average about 12 hours on the trip; and to any one who had to make it, the distance certainly seemed much longer than 35 miles. The other route started from Buena Vista in the Arkansas valley (for a long time the terminus of the D. and R. G. railway), thence passed either over Cottonwood pass (11,900 feet high), at the head of Cottonwood creek, or over Chalk Creek pass (12,300 feet high), into Taylor's park; thence

up the Taylor river and across Taylor pass (11,800 feet) at the head of it, to Castle creek and the mining town of Ashcroft; thence down Castle creek to Aspen. The distance from Buena Vista via this route is stated at 72 miles. Later on, the South Park railroad built a branch up Chalk creek to St. Elmo and the Mary Murphy mine, and stages were run between St. Elmo and Aspen. This route necessitated the crossing of two ranges, and the distance was longer, but the roads were generally better. Most of the freighting, however, was done via Hunter's pass, to and from Leadville and Granite. The first ore from any Aspen mine was shipped in the early spring of 1881, from the Chloride mine. It was packed to Leadville on "burros" or "jacks," and as much as $50 per ton was paid for the transportation alone. It goes without saying, that only very rich ore could be shipped in those times. The first road into Aspen over Taylor pass was built in the winter of 1880–81 and in the early summer of 1881. The road across Hunter's pass was built in the fall of 1881 and the spring of 1882. The first freight-rates from Buena Vista to Aspen were 6 cents per pound. But competition in freights and dulness of trade in the camp soon brought rates down to 2½ cents per pound, over either route; and before the advent of the railroads in 1887, freight had fallen to 1½ to 1¾ cents per pound, which is not much above what the railroads charge at the present time. The freight-rates on ore, which were to Granite from $15 to $20 per ton, constituted the main drawback to the camp's prosperity, being prohibitory of the mining and shipping of ores of lower grades. All such ores were thrown on the waste-dumps of the mines, which often consisted almost altogether of such ore, too low in its silver contents to stand the combined charges for freight and treatment. Tens of thousands of tons of such ore had accumulated on the dumps of the older mines when the railroads reached the camp; and a large part of the ore-shipments during the period from December, 1887, until now (April, 1888) has been taken bodily from these dumps, only the larger pieces of barren limestone being thrown out, and all the remainder shipped as ore.

The first mine discovered on Aspen mountain was the Spar mine on Spar ridge, on the west side of Spar gulch. This ore carried a large proportion of baryta, whence the name. Later discoveries were the Smuggler, at the foot of Smuggler mountain, and the Chloride on Aspen mountain, farther up Spar gulch than the Spar mine. A pocket of very rich ore was found here, cropping out on the sur-

face; and the product from this pocket was the first ore shipped from Aspen. Many other discoveries were made, but mostly of ore of too low a grade to leave much profit, after mining and transportation had been paid for. The town of Aspen was very "dead," when the discovery of the rich and large ore-body in the Emma shaft, soon followed by the intersection of the same ore-body in the Aspen shaft, set the camp "booming." The "boom," however, was of short duration; for these rich strikes led to a protracted "apex-side-line" litigation, only recently settled by compromise.

But the proof thus furnished of the existence of vast quantities, not only of low-grade ores, but also of high-grade silver-ores, finally brought the long-desired and hoped-for building of the railroads into Aspen. The Colorado Midland commenced work on its road at Colorado Springs in 1886, with the avowed aim of reaching Leadville and Aspen, and connecting both mining towns at the earliest possible date. In less than two years this broad-gauge road was completed, over the roughest country any broad-gauge road was ever built over, to Aspen, the first train running to Maroon creek, two miles below Aspen, in December, 1887, and into Aspen itself in February, 1888.

The building of the Midland road stirred up the Denver and Rio Grande company. While the Midland reached Aspen from Leadville by tunneling through Mount Massive, descending Frying Pan creek and ascending the Roaring Fork, the Denver and Rio Grande extended its already existing Red Cliff branch, from Leadville, via the Tennessee pass (the best railroad pass in Colorado) to Red Cliff, thence down the Eagle river to its junction with the Grand, down the Grand to its junction with the Roaring Fork at Glenwood Springs, and thence up the Roaring Fork to Aspen. Although occupying by far the longer route of the two, the Denver and Rio Grande was the first to complete its track into Aspen, reaching the town with its first train on October 28th, 1887.

Since then the town of Aspen has flourished. To one looking down upon it from Aspen mountain, new houses everywhere display their shining fresh lumber or bright paints. Solid brick business blocks are taking the place of the old log-cabins and slab-shanties. The population has about doubled since the advent of the railroads, and Aspen is to-day not only the most beautiful, but also the most prosperous of all the mining towns of Colorado. This prosperity is not ephemeral, but rests on the solid foundation of the mines, which in extent may not be able to compete with Leadville, but

which in the average richness of their ores far surpass the great Carbonate Camp.

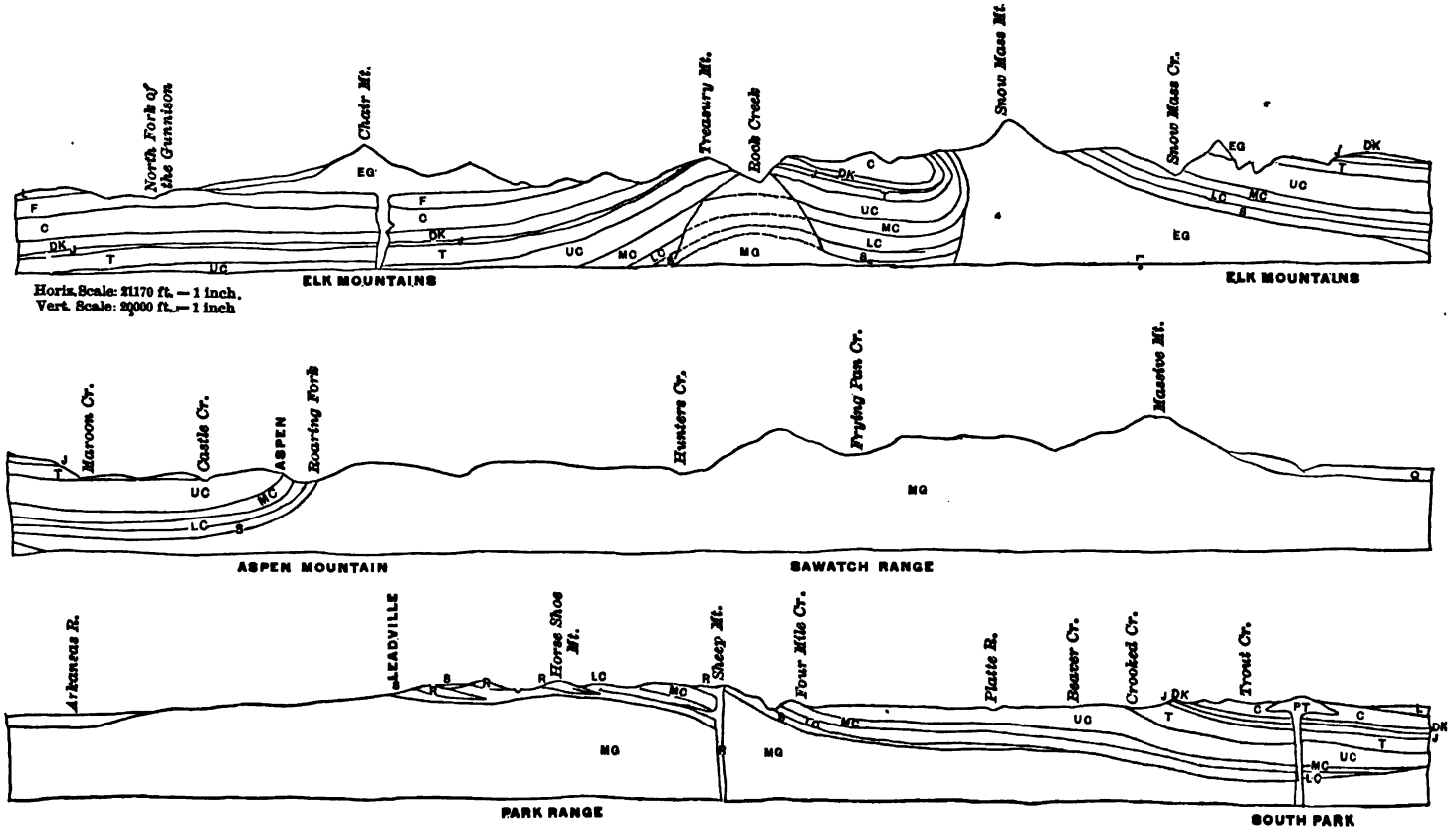
Much difference of opinion has existed and still exists concerning the nature of the ore-deposits of Aspen mountain. One main reason for this difference hitherto has been the intense bitterness of feeling between the parties to the "apex-side-line" law-suit, which naturally tended to disturb the accuracy of the observations made in the interest of either party. Although engaged on the "side-line" part of the suit, I believe, that it has had no influence whatever on the correctness of my observations and studies of the real nature of these ore-deposits. Entering on my investigations without any bias or prejudice either way, I have collected the facts presented below, and I trust that the conclusions, which I shall draw from them, will be such only, as they will fairly warrant.

The weekly shipments from Aspen are at present more than 2000 tons of ore. While much of this ore is still derived from the waste-dumps of the old mines, the shipments from this source are decreasing, while the total shipments are increasing, showing that the opening of the mines will more than compensate for the falling off due to the exhaustion of the dumps. Three hundred tons of ore per day will not be too large an estimate for the ore to be mined and shipped during this year. The value of the output of the camp will be largely increased by the working of the very rich ore-body in the Aspen mine, which has lain in sight, but untouched, for years, on account of the (now compromised) law-suit about its ownership.

II. General Geology.

That part of the Sawatch range which interposes its masses of Archæan granite and gneiss between Leadville and Aspen bears, on a map, nearly the shape of a crescent, with the concave side towards Leadville. This is mainly owing to the eastward bend taken by this range north of its highest peak, Mount Massive. While the general trend of the Sawatch range is north and south, the part of it between Leadville and Aspen has an axis trending northeast and southwest. Aspen and Leadville are about equidistant from the centre-line of the Sawatch range, assuming that this line passes through Mount Massive. As the sedimentary rock-formations at Aspen, however, dip at a much steeper angle than the corresponding formations at Leadville, we may be justified in assuming that the axis of the up-

FIG. 2.



Section from West to East along line DD, Fig. 1. (From Hayden's Geological Survey.) The section extends considerably beyond the limits of the map in Fig. 1. For explanation of geological symbols, see Fig. 1. The positions of Aspen and Leadville are given approximately.

heaval was situated in closer proximity to Aspen than to Leadville, *i.e.*, that this axis passed west of Mount Massive. (See Fig. 2.)

The town of Aspen is situated close to the boundary between the Archæan metamorphic granite on the east and the Palæozoic sedimentary rocks on the west. Crossing Hunter's pass and following down the valley of the Roaring Fork, we see on both sides the mountains composed of granite, until we emerge from the narrow valley into the flat, on which Aspen is built. Here at the upper (east) end of town, on the east side of Spar gulch, appears a bold outcrop of white quartzite.

These layers of Cambrian white quartzite rest immediately on the granite. They are the exact counterpart of the Leadville lower quartzite, beautifully white, coarsely crystalline, and containing, towards the top of the series, some calcareous layers, in which small deposits of rich ore have been found.

Above these quartzite beds, which are about 180 to 200 feet thick, are found calcareous and siliceous shales and thinly-bedded siliceous dolomitic limestones, apparently conformable with the quartzite. It is doubtful whether the beds of thin siliceous shales, containing some lime and magnesia, which lie between the siliceous dolomitic limestones of this group and the white quartzite, should be classed with the lower or the upper group.

The limestones, however, are evidently the counterpart of the Leadville Silurian white limestone. They have a similar light color, buff when weathered, and pale greenish or blue on their undecomposed fresh surfaces. The top layer of these Silurian limestone beds is formed by a rather thick bed of very homogeneous fine-grained almost lithographic dolomitic limestone. The thickness of these Silurian beds is about 180 feet. The upper layer, just referred to, underlies the lower of two parallel bands of quartzite which form a prominent geological feature among the rocks of Aspen mountain. The lower of these bands of quartzite has a thickness of nearly 7 feet. It consists mainly of a homogeneous glassy mass of quartz-rock with yellowish brown enclosures, probably of oxides of iron, all through it, which give a buff color to the whole mass. It is separated from the upper band of quartzite by 23 to 24 feet of siliceous dolomite layers, of the same appearance and character as the Silurian dolomites below. The upper band of quartzite, 9 feet thick, is from the reddish color, which it has on its weathered surface, one of the most prominent and easiest recognized layers of rock on Aspen

mountain, and forms a convenient datum-line for measurements. This quartzite band presents rather the character of a fine conglomerate of quartz-grains, enclosing between them some ferruginous matter, which gives to the rock on fresh surfaces a yellowish-brown tint, turning red on weathering. The two quartzites with the enclosed 24 feet of siliceous dolomites, in a total thickness of 40 feet, agree in this feature, and in their geological position, with the 40 feet of "Parting quartzite," which in Leadville, according to Emmons's researches, divides the Silurian from the Carboniferous formation.

Above these quartzite bands, which I shall hereafter call the parting-quartzites, we find the Lower Carboniferous limestone, which, at Aspen, as at Leadville, and in many other western localities, is the ore-bearing rock *par excellence*. These Lower Carboniferous limestones, on Aspen mountain, are from 300 to 400 feet thick, more heavily bedded than the Silurian dolomites, and contain, where unaltered, much less silica.

This limestone occurs either as a dolomite or as a pure limestone. The first has received the local name of "brown" limestone; the latter, that of "blue" limestone, from their prevailing colors, as found *in the mines*.

The Lower Carboniferous limestone of this region has been partly dolomitized. The mass of dolomite in this formation forms always the lower layers, while the upper 100 to 150 feet are mainly unaltered limestone. These upper layers of "blue" limestone contain, however, local bands, lenticular masses and irregular bodies of dolomite, undistinguishable in character, composition and appearance, from the dolomite forming the lower layers of the same formation.

There are many reasons for the assumption, that these dolomite layers were not originally deposited as such, but were deposited as pure limestones, and afterwards partly dolomitized, and that this dolomitization is still going on.

The first reason is, that, while the total thickness of this formation remains reasonably constant over large areas of country, the division into dolomite and blue limestone is only local, *i.e.*, the relative thicknesses of the underlying body of dolomite and of the overlying blue limestone vary very much in different localities. I am informed that this change in the thickness of the dolomite and blue limestone can be very clearly seen on the face of a bluff, which follows the course of Rock creek for nearly five miles, and is entirely com-

posed of this Lower Carboniferous limestone, where the thickness of the dolomite dwindles to 15 or 20 feet and swells to over 100 feet, with a corresponding increase and decrease in the thickness of the overlying blue limestone. But we need not go outside of Aspen mountain to verify this statement. The thickness of the blue limestone is found to vary here between 90 and 150 feet.

The second reason for assuming a dolomitization of originally pure limestone, is the fact that the division-line between the underlying dolomite and the overlying blue limestone, the so-called "contact," does not always follow a clearly-defined seam, or even a stratification-plane. Where the "contact" is found in undisturbed position in the mines, we frequently see the division between blue and brown limestone pass through the middle of a solid layer of limestone, the upper part of this layer being typical unaltered blue limestone, and the bottom typical dolomite, and the one changing into the other along an indefinite curved or zig-zag line, which makes it difficult to say, where the dolomite begins and the blue limestone ends. I have seen hand-specimens taken from this so-called "blended contact," which, although not over two inches thick, showed the division-lines between blue and brown limestones on the opposite faces of the slab, at right angles to each other. Besides, as already stated, the lenticular bodies and irregular masses of dolomite, occurring irregularly and locally in the upper mass of unaltered Lower Carboniferous blue limestone, are of exactly the same composition, texture and appearance as the dolomites from the lower beds.

Some of the richest ore-bodies on Aspen mountain having been found near the junction of the dolomitic and the blue limestone of this formation, it has been maintained that this dolomite was of Silurian age, and the "blue" limestone only Lower Carboniferous; that the ore-deposits occurred as a contact-vein between the two formations. In view of the above-cited facts, I cannot accept this theory. There is no doubt that, at least, the upper mass of the dolomites above the parting-quartzite bands, and probably all beds above these quartzite bands, belong to one and the same geological age, which we may safely pronounce to be Lower Carboniferous, since the unaltered blue limestone of this formation is, without doubt, of that age.

Professor Lakes, in his report on *The Geology of the Aspen Mining Region*, in the Biennial Report of the Golden State School of Mines, 1886, assigns these dolomite beds, together with those below

the parting-quartzites (which latter he mentions as a bed of brown quartzite, 20 to 50 feet thick, occurring in the dolomite beds), to the Silurian or the Sub-carboniferous formation, or partly to both. He is, also, of the opinion that the dolomites were originally deposited as limestones, and only afterward changed to dolomites. An essential distinction between the dolomites above and those below the parting-quartzite bands escaped the observation of Professor Lakes. While the silica-content of the Lower Carboniferous dolomites is very low and corresponds to the silica-content of the unaltered blue limestone of the same formation, all the dolomites below the parting quartzites contain much silica—never less than 10 per cent.

This difference in silica is one of the characteristic distinctions between the Lower Carboniferous dolomite of Leadville, the ore-bearing rock of that region, and the Silurian white limestones below the parting quartzite.

Another distinction between the Lower Carboniferous dolomites and the Silurian dolomites of Aspen, is, that the former are, on an average, much more heavily bedded than the latter. In fact, these dolomites are of such different lithological character, that, after getting acquainted with their appearance, there is no danger of mistaking one for the other.

Above the Lower Carboniferous formation, overlying the "blue" limestone beds, we find on Aspen mountain, dark blue or black calcareous shales, often exceedingly siliceous, and always very pyritiferous. These correspond to the Weber shales overlying the blue dolomitic limestones of Leadville. Among these shales, as in Leadville, we find local seams of impure coal and much carbonaceous matter, which gives to the rocks of this formation their dark, sometimes jet black color. There are, however, among these shales, local developments of limestones much resembling and frequently mistaken by the miners for the desired blue limestone of the lower formation. These Middle Carboniferous limestones, however, carry always more or less pyrites, as an impregnation and in seams, while the Lower Carboniferous limestone is almost entirely free from pyrites. The Middle Carboniferous limestones, moreover, contain a greater amount of silica and from 10 to 15 per cent. of carbonate of magnesia. The thickness of this formation is hard to determine, as it is very seldom found in its entire thickness in one place, much of it having been removed by erosion, and it is also

split up in wedge-shaped or lenticular masses in most places by intrusive sheets of eruptive rock. The Last Scheme shaft on the north slope of Aspen mountain, above the town, has passed vertically through more than 300 feet of these shales, and is still in them. It is therefore safe to assign to the Weber shales at this point a thickness of over 300 feet.

This completes the description of the sedimentary rocks found on Aspen mountain. For the sake of comparison, I give the geological section of the sedimentary rocks of the Mosquito range (Leadville) according to Emmons; that of the Aspen mountain rocks as given by Prof. Lakes in his report; and, finally, my own section of the latter.

Leadville Section (Mosquito Range; Emmons).

Geological Age.	Description of Rocks.	Thickness.
Carboniferous. 550 feet.	Weber shales, Middle Carboniferous	350
	Compact, heavily bedded, dark blue limestone, a typical dolomite, with black chert on top layer. "Blue" limestone. Lower Carboniferous	200
Silurian. 160 feet.	White quartzite. Parting quartzite. Silurian	40
	Light gray siliceous dolomitic limestone with white chert concretions. "White" limestone. Silurian	120
Cambrian. 200 feet.	White quartzite passing up into calcareous and argillaceous shales. Cambrian	200
Archæan.	Archæan granite.	

Aspen Mountain Section (Prof. Lakes).

Carboniferous. 180 feet.	Weber shales, black, calcareous, pyritiferous, and coal-seams. Middle Carboniferous	60
	Lower Carboniferous, "blue" limestone, compact, heavy bedded, bluish-gray limestone, not dolomitic, with siliceous concretions of black chert	120
Silurian. 340 ft.–370 ft.	Drab dolomite brecciated near top	100
	Brown quartzite	20
	Drab and pale gray dolomite with white chert	200
	Brown quartzite (position uncertain)	20–50
Cambrian. 300 feet.	Light gray, very siliceous limestone, dolomitic with white chert concretions	100
	White quartzite, some thin shales on top and stratum of siliceous limestone	200
Archæan.	Archæan granite.	

Aspen Mountain Section (C. Henrich).

Over 650 feet of Carboniferous, of which the lowest layers may be Devonian.	Middle Carboniferous Weber shales. Dark blue, often black, calcareous slates, often very siliceous, always pyritiferous, with local developments of dark blue limestone beds and seams of impure coal.....	over 300
	Lower Carboniferous limestone. Dolomite in its lower, pure limestone in its upper layers, the latter containing local bodies of dolomite.....	350
40 feet Devonian. (?)	Reddish quartzite band of fine conglomerate texture.....	9
	Fine-grained siliceous dolomite.....	24
	Buff-colored quartzite.....	7
Silurian. 160-180 feet.	Light-colored, thinly-bedded, siliceous dolomites.....	160-180
Cambrian. 200 feet.	White quartzite beds, with thin siliceous shales and calcareous beds on top.	200
Archæan.	Archæan granite.	

It will be seen from the above compilation, that the main difference between myself and Prof. Lakes is to be found in his assigning the lower siliceous dolomites to the Cambrian series, while I find in them the counterpart (almost identical) of Emmons's Silurian white limestone of Leadville. Then Prof. Lakes assigns to the parting quartzite an uncertain position, and gets a greater thickness of the strata above it. This stratum of brown quartzite, to which he assigns a thickness of 20 to 50 feet, is unknown to me, and I am led to belie e, that, in getting his geological section on the slopes of Spar gulch, he has unknowingly counted one of the quartzite bands and some of the siliceous dolomites twice. This may very easily occur in that region, if no account is taken of the numerous breaks or faults, which run obliquely to the strike of the strata and displace the outcrops of the different rocks, so that one and the same quartzite band may easily appear as two different ones, occurring in different geological horizons. These faults had been scarcely developed, and were unrecognized at the time of Prof. Lakes' examination.

The eruptive rock of Aspen mountain, called by the miners porphyry, has been so much decomposed throughout its mass, that even from a depth of 200 feet below its top and the surface, specimens taken from the Schiller shaft and analyzed by Mr. A. W. Hare contained 9 per cent. of carbonate of lime. Prof. Lakes's report contains the result of the microscopical examination of pieces of the "porphyry" of this region by Prof. Wendell Jackson, University of California, which I give here:

"The rocks are so decomposed that determination is difficult.

Aspen mountain 'porphyry' contains quartz in great abundance, with liquid inclusions and movable bubbles in the quartz. The feldspar may be either plagioclase or orthoclase (too decomposed). Chlorite occurs as a secondary production in fibrous aggregates, assuming the columnar form of the original mineral from the decomposition of which they have resulted. That mineral is probably hornblende. It is long, columnar, has oblique extinction of 25°, with columnar axes, but without pleochroism, so far as I can detect, which hornblende generally has. The rock is probably a quartz-diorite. The rock from Richmond Hill contains orthoclase and plagioclase feldspar, porphyritically disseminated through a fine-grained base of quartz and orthoclase with large crystals of quartz."

The "porphyry" occurs in intrusive sheets, and as dykes through the sedimentary rocks. The main mass of the porphyry left on Aspen mountain (the greater part having evidently been long ago removed by erosion) occurs as a thick intrusive sheet in the Weber shales, only a small thickness of these shales generally intervening between the top of the Lower Carboniferous limestone and the porphyry. It is, however, found in intrusive sheets and in dykes cutting diagonally across the strata, in nearly all the sedimentary rocks of the mountain. There are two varieties of this porphyry, the one, evidently the older, forming the main intrusive sheet now constituting the top of Aspen mountain. It contains very little pyrite, distributed in extremely fine grains through its mass. It is white with a slight greenish hue, due to the chloritic particles contained in it. It has, like the Leadville white porphyry, the characteristic, that when weathered, it is stained, from the surface in, by concentric layers of iron oxide.

The second variety occurs chiefly as dikes breaking through the stratified rocks, and to a much smaller extent than the former. It has more of a dark gray to gray-and-white mottled color, and contains a large amount of pyrites, sometimes as high as 60 per cent. of its mass being composed of that mineral (this, of course, only in spots). It is very easily decomposed, and, when totally decomposed, forms a plastic, tough clay, containing a large amount of quartz sand and pyrites. These clays vary in color from dark gray to white, and are frequently found in some of the mines. Other products of this decomposition are: a honeycombed quartz-rock containing a large amount of pyrites, and a coarse, gritty quartz-sand, loosely cemented together. Where this porphyry comes in contact with sedimentary rocks, the parts near the contact are frequently changed to quartz-rock.

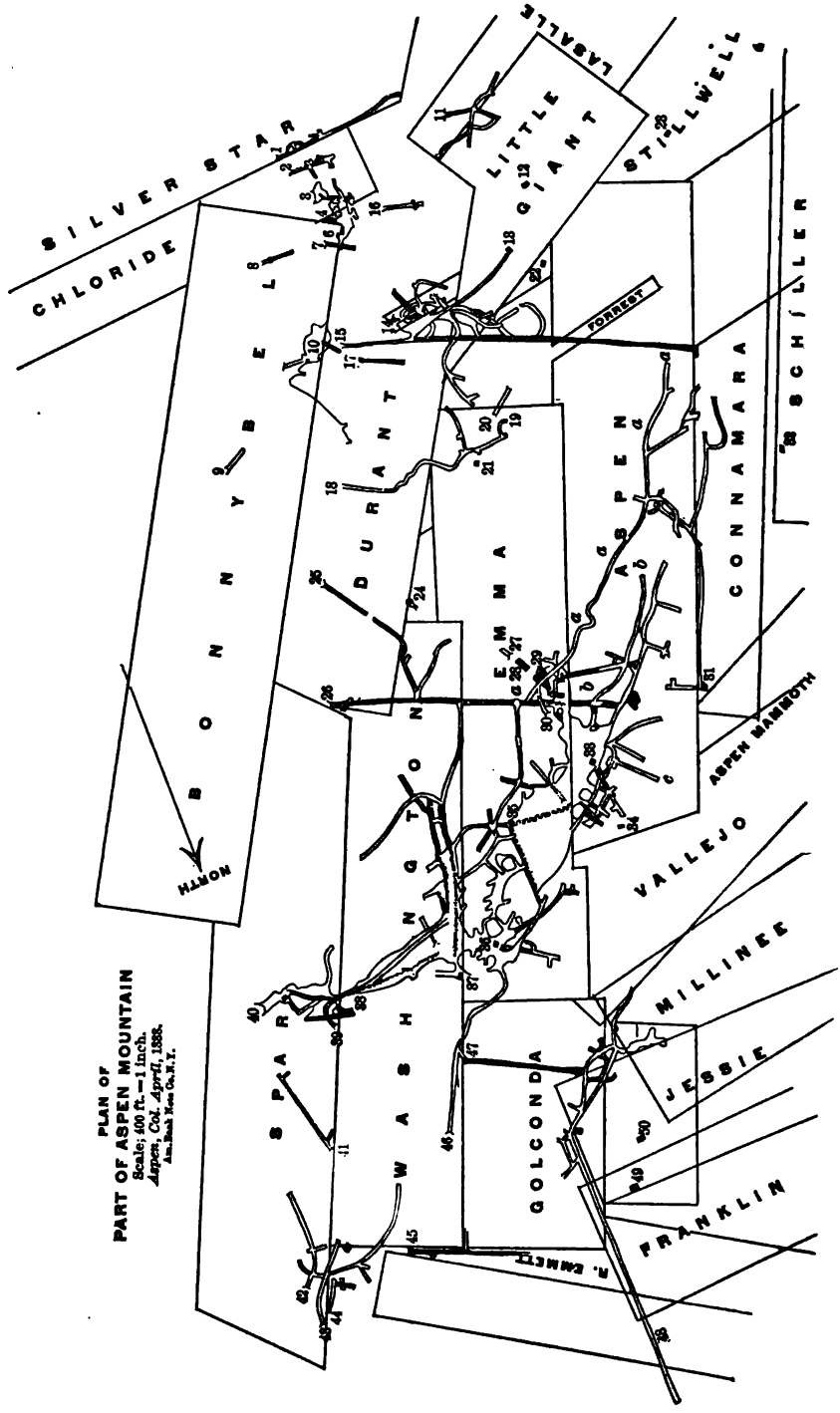

PLAN OF
PART OF ASPEN MOUNTAIN
Scale; 400 ft. = 1 inch.
Aspen, Col. April, 1888.
Am. Book Mine Co., N.Y.
SILVER STAR
CHLORIDE
BONNYBELL
LITTLE GIANT
LASALLE
STILLWELL
SCHILLER
FORREST
DURANT
EMMA
ASPEN
CONNAMARA
ASPEN MAMMOTH
VALLEJO
MILLINEE
JESSIE
FRANKLIN
GOLCONDA
WASHINGTON
SPAR
NORTH
R. EMMETT

FIG. 3.

III. Special Geology.

Aspen mountain presents, in the small area which it covers, one of the most complicated geological structures imaginable. As already observed, it is bounded north by the flat, on which the town of Aspen is built, and whence it rises precipitately 2000 feet to the crest visible from town, thence on a more gentle slope to Turtelotte park, which separates it from the southern continuation of the mountain spur between the waters of Difficult and Castle creeks, parallel tributaries to Roaring Fork. On the west it is bounded by the valley of Castle creek, and on the east by Spar gulch, which in its upper course gets a little farther away from Castle creek, thus affording a greater breadth to the higher part of Aspen mountain. About midway between Spar gulch and Castle creek, running up the north slope of the mountain, as will be recalled from the description already given, Pioneer gulch divides it into Aspen mountain proper on the east, and West Aspen mountain on the west.

Fig. 3 is a plan of part of Aspen mountain, indicating the mine workings. On this plan the different workings are numbered, as follows:

1. Silver Star, old workings.
2. Chloride south incline.
3. Chloride north incline.
4. Grand Opening shaft (since relocated and renamed).
5. Durant-Chloride workings.
6. Old Bonnybel shaft.
7. Bonnybel south incline.
8. Bonnybel discovery-tunnel.
9. Bonnybel "parting-quartzite" tunnel.
10. Eames's Bonnybel workings.
11. La Salle shaft and workings.
12. Teaser discovery-shaft.
13. Teaser working-shaft.
14. Little Giant workings.
15. Visino tunnel and incline.
16. Apex tunnel No. 14.
17. Apex incline No. 13.
18. Apex tunnel No. 8.
19. Forrest black shale shaft.
20. Forrest west tunnel.
21. Forrest main shaft.
22. Democrat shaft.
23. Stillwell shaft.
24. Delta shaft.
25. Apex incline No. 4.
26. Durant incline.
27. Emma discovery-tunnel.
28. Hutchinson's Emma incline.
29. Emma "Cave" shaft.
30. Emma tunnel.
31. Connamara shaft.
32. Schiller shaft.
33. Aspen shaft.
34. Hercules shaft.
35. Emma shaft.
36. Vallejo shaft.
37. Washington No. 2 shaft.
38. Washington shaft.
39. Spar working-incline.
40. Spar discovery-tunnel.
41. Spar brown-lime tunnel.
42. } Lower Spar workings.
43. }
44. General Shield's tunnel.
45. Robert Emmett shaft and incline.
46. Golconda tunnel.
47. Golconda shaft.
48. Veteran tunnel.
49. Franklin shaft.
50. Jessie shaft.

a a a. Durant-Aspen "court drift."

b b. Aspen No. 2 drift.

c. West cross-cut in Aspen 2d level.

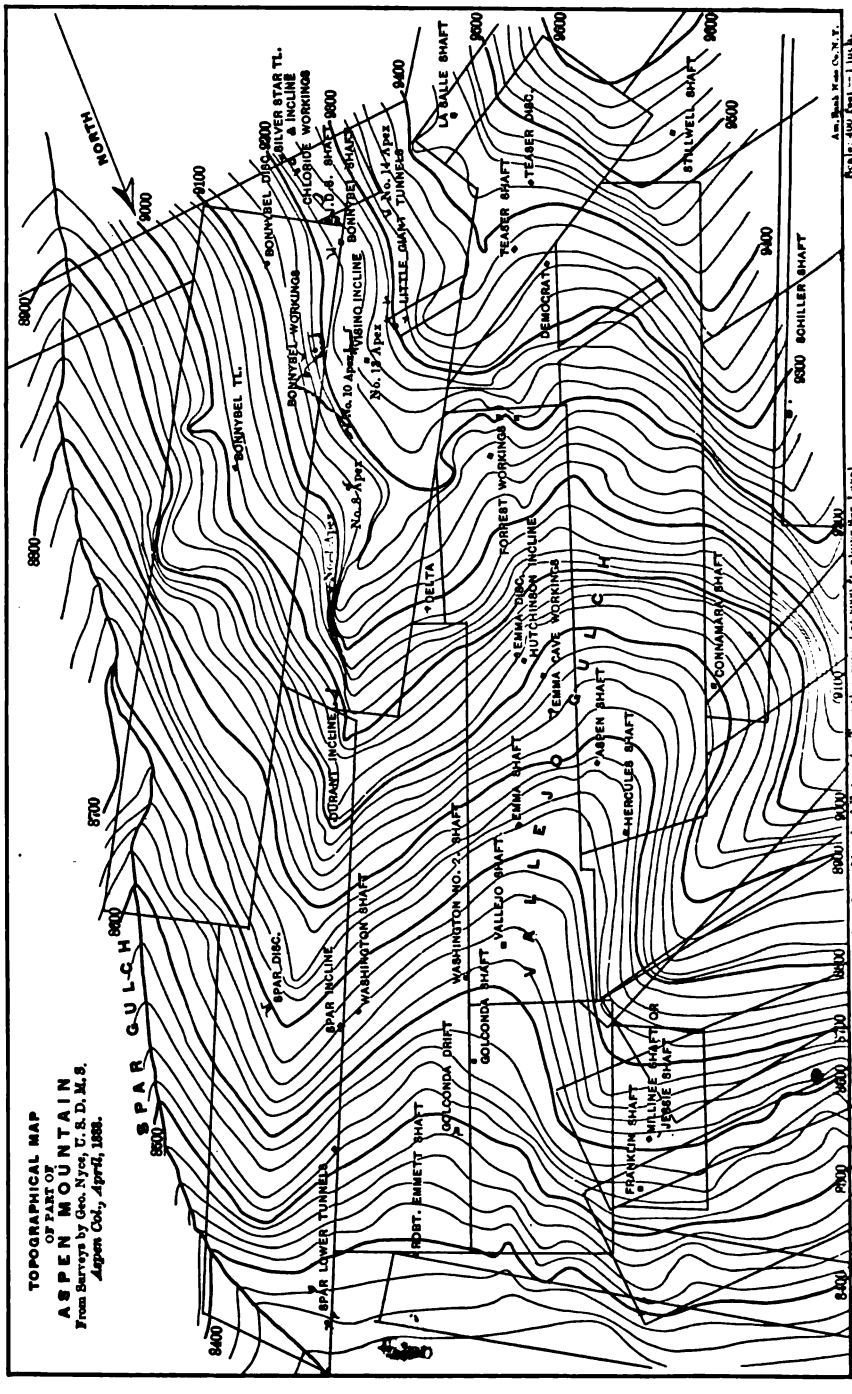

TOPOGRAPHICAL MAP
OF PART OF
ASPEN MOUNTAIN
From Surveys by Geo. Nyce, U.S.D.M.S.
Aspen Col., April, 1881.
NORTH
SPAR GULCH
VALLEJO GULCH
BONNYBEL TL.
BONNYBEL DISC.
BONNYBEL WORKINGS
SILVER STAR TL.
& INCLINE
CHLORIDE WORKINGS
U.S.A. SHAFT
BONNYBEL SHAFT
No. 10 Apex
No. 8 Apex
No. 12 Apex
No. 14 Apex
LITTLE GIANT TUNNELS
LA BALLE SHAFT
TEASER SHAFT
TEASER DISC.
DEMOCRAT
STILLWELL SHAFT
SCHILLER SHAFT
DELTA
FORREST WORKINGS
EMMA DISC.
HUTCHINSON INCLINE
EMMA CAVE WORKINGS
EMMA SHAFT
ASPEN SHAFT
HERCULES SHAFT
CONNAMARA SHAFT
VALLEJO SHAFT
WASHINGTON SHAFT
WASHINGTON NO. 2 SHAFT
GOLCONDA SHAFT
GOLCONDA DRIFT
ROBT. EMMETT SHAFT
SPAR INCLINE
SPAR DISC.
SPAR LOWER TUNNEL
FRANKLIN SHAFT
WILLIAMS SHAFT OR
JESSIE SHAFT
Contour Lines 50 Feet Vertical Apart.
Base Line Mouth of Enterprise Tunnel, assumed at 9000 ft. above Sea Level
Scale 400 Feet = 1 Inch.

FIG. 4.

To get a tolerably clear notion of the geological features of Aspen mountain, one should ascend Spar gulch to the Princess Louise shaft, on the face of its eastern quartzite slope, about a mile above town; thence ascend the western slope of the gulch to the top of the upper end of Spar ridge, between the Little Giant discovery cut and the La Salle shaft, past the Bonnybel, Chloride and Visino mines; thence cross the head of Vallejo gulch, past the Aspen and Connamara shafts to the Schiller shaft, on the crest of the ridge forming the west side of Vallejo gulch; thence follow the county-road to the head of Pioneer gulch on the top of Aspen mountain, inspecting the shafts and tunnels on the west side of that gulch; then follow along the steep northern slope of West Aspen mountain to its end near Castle creek at the west end of the town. This route will be pursued in the following description. It might be partly represented on the map, Fig. 4, by drawing an irregular curve as follows: Beginning where the 8740-foot contour-curve crosses Spar gulch, passing the Bonnybel tunnel and workings, the Visino incline, the Little Giant tunnels, the Teaser shaft and the Democrat, to the Schiller shaft, near the end of the 9300-foot contour-curve.

The east side of Spar gulch, below the Princess Louise shaft, is composed altogether of the white layers of the Cambrian quartzites. The small amount of rich ore so far produced by this mine has come from the quartzites, or probably more correctly, from one of the calcareous layers intercalated among them. This, at least, is the supposition, indicated by the location of the shaft and the character of the ore. I am not acquainted with the workings of the mine. It was worked steadily during last winter, but is now closed. The ore is said to be rich, and looks rich; but the quantity produced has been small. The bed of the gulch marks approximately the upper limit of the Cambrian quartzites; at least the upper layers of this formation do not extend far up the western slope of the gulch. The stratification of the quartzite here is very plain, many of the denuded edges forming the surface of the hill on the east slope of the gulch. The dip of the Cambrian strata here is N. 30° W. 40°, which is about the prevailing average dip of the sedimentary formations in the vicinity of Aspen.

Ascending the west slope of the gulch, we pass over *detritus*, mainly of the Silurian dolomites, and soon reach the foot of a vertical bluff of these, showing their thinly-bedded layers to the height of about 80 feet, and dipping in the same direction and at the same angle as the quartzite beds just described. Above this bluff is the

Bonnybel Quartzite tunnel, so called because it follows the lower parting quartzite for some distance along its strike into the hill. A small amount of very rich ore, the remaining traces of which are still to be seen, was taken in early days from this tunnel. The upper band of the parting quartzite crops out a little distance above, and may be followed obliquely S. W., up the hill, to its end in a small cut, directly under the waste-dump of the Bonnybel mine. Here a cross-fault throws the formations on the south about two hundred feet lower than those on the north of it. The course of this fault has been shown by surface-strippings and cuts, made by the "side-line" people during the late litigation. This work led to the discovery, in one of the cuts south of the fault, by Mr. —— Eames, who conducted these surface-explorations, of the outcrop of the famous Bonnybel ore-body. He secured a six months' lease on the Bonnybel claim, then thought almost worthless. Over $600,000 worth of ore was shipped from this mine, or rather quarry, during last winter. The mining operations consisted almost exclusively in excavating a huge chamber in the side of the mountain, the thin roof of rock and earth left between this chamber and the outside being cemented and held firmly by the frost. Nearly all the work was done during the hours of the day, in daylight, and almost the whole of the material excavated was shipped as ore. When spring came and the frost left the ground, the roof of the chamber tumbled in and gave the mine more than ever the appearance of a large quarry.

From the small cut which exposes the fault, cutting off the quartzite band, we ascend to the Bonnybel mine and the Visino tunnel directly above it, passing first over Lower Carboniferous dolomite for about 50 feet (to which thickness, however, should be added the 200 feet of throw by the fault, since we now ascend on the south side of the fault, while before we crossed the end of the quartzite band we were on the north or relatively uplifted side). The upper part of these 50 feet of dolomite is marbleized, and the cause may be seen in the tongues of porphyry penetrating the marbleized limestone in many parts of the Bonnybel workings. From this dolomitic marble we pass to marbleized blue limestone on top of the Bonnybel workings, and keep on this kind of rock, until we reach the Visino shaft-house. All the rocks thus far passed over dip northwesterly at the same, or nearly the same, angle as the quartzite in the bed of Spar gulch.

The site of the Visino shaft-house has been excavated in a body of

porphyry, which here comes up through the marbleized limestone. This porphyry ends abruptly to the northward in those surface-cuts which led to the discovery of the Bonnybel ore-body. Its lower side, here in cut No. 10, pitches S.W. at an angle of 35° from the horizontal. Immediately below the Visino shaft-house a short tunnel has been driven through the marbleized blue lime to the porphyry. Here the pitch of the contact between marble and porphyry is nearly vertical, and a drift carried 20 feet along this contact shows no alteration in this pitch. The porphyry is changed for about a foot from the contact into a quartz rock, impregnated with much pyrites, while near the contact seams of baryta run through the marble as well as through the porphyry. The marble in the immediate vicinity is slightly impregnated with silver, assaying in spots 3 to 4 ounces to the ton.

Beyond a belt of porphyry about 50 to 60 feet wide is the blue limestone bluff of the Little Giant mine. In the face of this bluff are two bands of dolomite, each about 3 to 4 feet thick. All these rocks are still dipping N.W. This limestone bluff, the most northerly of many of like appearance, projects like a tower from the eastern slope of the ridge running southerly to Turtelotte park. On the top of the bluff is the Little Giant discovery cut, made in the blue limestone, on a vertical seam of spar and ore.

In passing along the top of this limestone knoll, there is seen to the south porphyry cropping out on the surface; and beyond this, about 350 feet to the south, another limestone bluff springs from the slope of the mountain, behind which bluff, to the west, appears the La Salle shaft-house, west of which, again, the porphyry comes to the surface and forms the rest of the mountain-side to its summit.

On the north, the limestone knoll forms a cliff, falling down vertically almost to the Forrest tunnel, driven at the bottom of the cliff for some distance into solid blue limestone. At the mouth of this tunnel is a small tongue of porphyry, very much decomposed, pitching steeply south under the bluff. This porphyry tongue marks the extension of either the Bonnybel fault, or another fault developed in the workings of Apex tunnel No. 8 (see the topographical map of this part of Aspen Mountain, Fig. 4), or, most probably, the continuance of both. That a large fault runs through here, is further shown by one of the Forrest shafts, which has been sunk close to the mouth of the tunnel, and below it, in Middle Carboniferous black shale.

Looking N.E., down Vallejo gulch, is seen the shaft-house of the

Golconda shaft, sunk through the Middle Carboniferous shale into the Lower Carboniferous, and connecting, with an inclined winze over 400 feet deep, the Spar, Washington, Emma, and Vallejo mines, belonging to the Aspen Mining and Smelting Company, with the Veteran tunnel. This winze is now used as an ore-chute and tramway to deliver the ore of the mines mentioned at the mouth of the Veteran tunnel, only a few hundred feet above the track of the Colorado Midland railroad, whence it is brought cheaply, by a system of inclined and gravity tramways, into ore-bins near the railroad track, to be finally dumped through an inclined chute directly into the railroad cars. The interposition of receiving-bins between the mines and the cars is necessary to keep apart the different kinds of ore, which vary materially in their character and cost of treatment.

Above this Golconda shaft, and scattered over the western slope of Spar ridge, where the blue limestone crops out everywhere, appear the shaft-houses of the Spar incline, and of the Vallejo and Emma shafts. The Aspen shaft is located farther up in the bottom of the Gulch, and had to pass through 60 feet of porphyry "wash" and shales, before reaching the Lower Carboniferous limestone.

Still farther up, and on the west side of Vallejo gulch, is the shaft-house of the Connamara shaft, which was sunk 620 feet, the bottom being still in the Lower Carboniferous blue limestone. A large fault evidently passes between the Aspen and Connamara shafts, throwing the western side a considerable distance below the eastern. The broken country near this fault, if not the fault itself, has been encountered in the long S.W. drift driven by the Durant party under orders of the Court across the Aspen claim into the Connamara claim, and also at the bottom of the Visino incline, near the Connamara-Aspen boundary line, 750 feet below the surface. The dip of the fault at the points mentioned seems to be steeply east, while in the workings of the Veteran and Enterprise tunnels, as well as in the Robert Emmett incline, driven in a northwesterly direction, the breaks or faults encountered dip northwest. It is highly probable that either or both of the faults encountered in these workings are the northern continuation of the Connamara fault.

Northwest, towards the Schiller shaft, the eastern boundary-line of the outcrop of the Weber shales is crossed a little east of the Teaser shaft. This shaft, at a depth of 24 feet, leaves the slates and enters the blue limestone, which at this point has a vertical thickness of only 95 feet. The ground here is covered deeply with porphyry

wash, so that it cannot be determined at what point the Weber shales cease and the outcrop of the main porphyry sheet begins. The Schiller shaft was sunk 375 feet through solid porphyry to the Weber shales, from which point to its present depth of about 425 feet, it has passed through alternate layers of shales and white and dark-gray porphyry. When continued to the top of the Lower Carboniferous limestone, this shaft will afford a very interesting section, throwing much desirable light on the geological structure of this part of the mountain. The surface of the mountain, from the Schiller shaft west to the head of Pioneer gulch, is covered with porphyry, constituting the main intrusive sheet of the mountain. Of the maximum thickness of this intrusive sheet very little is now known. Its greatest observable vertical thickness is exposed at the head of Ophir gulch, about a mile south of the head of Pioneer gulch, where it breaks off in a vertical bluff, showing a vertical thickness at that place of 800 feet of porphyry.

Neither deep shafts nor any other developments of importance exist at present between the Schiller shaft and Pioneer gulch. All we know, thus far, is that this part of the mountain is covered by the remnants of the immense intrusive sheet of porphyry, the largest part of which has long since been removed by erosion.

The central part of West Aspen mountain, on the west side of Pioneer gulch, is formed by a sharp ridge of granite, which occupies the axis of a very steep anticline, all the sedimentary rocks and part of the granite itself on the summit of the anticline at the top of the central part of West Aspen mountain having been removed by erosion, if they were ever there. It is more probable that the strata of the sedimentary rocks were broken and torn asunder by the upheaval of the underlying granite. The direction of the axis of this anticline is north and south, and therefore crosses West Aspen mountain diagonally, afterwards running farther south, parallel with and west of upper Aspen mountain. As the crest of this uplifted ridge of granite seems to have been nearly horizontal, and as the whole formation rises to the southeast, the farther south we go on this summit of the ridge, the nearer we get to the original surface of the main Archæan granite body of the Sawatch range, and the less will its disturbing agency on the overlying sedimentary strata be observed. While the result of the upheaval of this granite ridge along the west side of Pioneer gulch has doubtless been a very sharp folding and final faulting of the sedimentary strata, it has resulted, to the south of the head of Pioneer gulch, in a gradually

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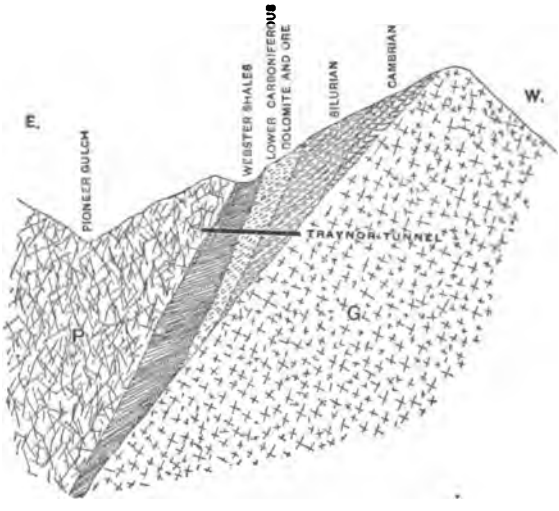
does not appear to have extended beyond the valley of the Roaring Fork. The red and white sandstones of Red mountain seem to overlie conformably the Lower Carboniferous rocks encountered in the mines of Smuggler mountain, and dipping regularly northwest about 35° from the horizontal. This seems to confirm the theory

FIG. 5.



SECTION ON WEST ASPEN MOUNTAIN.

that the valley of the Roaring Fork, near the town of Aspen, indicates the course of a large cross-fault running N. W. and S. E. beyond which the disturbance caused by the uplift of the West Aspen mountain granite has not been felt.

Castle creek itself marks without doubt the course of an immense fault, which throws the Archæan and Palæozoic beds on its west side at least 5000 to 6000 feet lower than on its east side. A few miles above the town of Aspen, on the east side of Castle creek, the Triassic red sandstone can be seen at the bottom of the creek, resting against the mountain side, while 2000 feet above the level of the creek the palæozoic rocks appear on top of the Archæan granite. This immense fault was caused evidently by the uplift of the Elk mountain range, after the Sawatch range had already tilted the sedimentary rocks to an inclined position. The result was a reverse folding, clearly discernible along the valley of Castle creek, and a final faulting about midway between the axes of both mountain-chains, the position, mainly occupied by Castle creek.

The result of these two upheavals on Aspen mountain are two main systems of faults, one running parallel to the strike of the sedimentary rocks from northeast to southwest and the other nearly at right angles to it, parallel to the axis of the Elk mountains, from southeast to northwest. The dip of the northeast faults is generally northwest, cutting obliquely through the strata at a steeper angle (on an average 60° from the horizontal), while the northwest faults (rather more north than west, in fact) generally differ, as well in their direction as also in their inclination, the dip varying at different depths of one and the same fault.

It thus appears that the geological structure of Aspen mountain is not quite as simple and regular as it has been shown to be on some published sections. One of these sections through Aspen mountain was published by —— Mitchell, C.E., in Denver, and is on exhibition in the Clarendon, the main hotel of Aspen. It has evidently been prepared with the sole purpose of showing a continuous contact-vein or lode, with an outcrop on the Durant claim, and reaching down with a gradually increasing dip to the innermost recesses of the earth. Such publications may serve their purpose, but they do an immense amount of mischief, by fortifying the already too prevalent notion of the continuous contact-vein character of these deposits, leading to useless expenditures of much money and work in the wrong direction in prospecting, and keeping people from prospecting and developing their claims in the right way.

I shall now proceed to a description of the main features of those mines on Aspen mountain with which I have become familiar.

IV. Principal Mines.

The Spar Mine.—Entering the Spar mine through the old discovery-cut and tunnel (No. 40 on general map of the mine-workings, Fig. 3), we observe at the mouth of the open cut leading to the tunnel a highly-fissured limestone, through the fissures and through the general wall of which run in all directions numerous smaller seams, veins, veinlets and impregnations of baryta, chiefly, however, following the bedding planes and the wide fissures and cracks which run parallel to the Spar fault, as shown along its dip in cross-section (Fig. 6) and also in the section through the Spar mine (Fig. 8). The ore, which is here a baryta ore, is found in the limestone, on both sides of a wide crevice, filled with decomposed and crushed material, resulting from downthrow of the west side of the Spar fault for about 100 feet, slope-measurement. This crushed

decomposed material, filling the fault-crevice, carries some lead-ore. The fault here, at the entrance to the Spar mine, is between dolomite limestone on the east as its foot-wall, and "blue" limestone on the west side as its hanging-wall. Both walls are impregnated with ore at this place. About 20 feet from the tunnel-mouth a winze has been sunk on the fault following the ore. Beyond this winze, to the southwest, all the rich ore has been stoped out above this level, leaving, however, plenty of lower-grade ore exposed. The winze followed down a rich ore-streak, and large quantities of high-grade ore were taken in the early times out of the levels run from this winze and from the stopes between those levels. (See plan of the mine, Fig. 7.) As the workings got deeper and the hoisting of the ore and waste through the winze and tunnel more difficult, a cross-cut incline was run in from the west side of Spar ridge through the blue limestone to what was then called the Spar vein.

The winze just mentioned, follows the fault down to a depth, where both sides are dolomite. This was rather a contradiction of the theory first held as to the character of the Spar vein as a "contact fissure-vein" between dolomite (supposed to be Silurian) and "blue" limestone (Lower Carboniferous). Still the theory was maintained, and the fact explained as a local branching off into the foot-wall. After the completion of the incline, however, the discovery and exploitation of the large and extremely rich ore-deposit called the Bridal Chamber, still farther away from the contact of the blue and brown limestones, ought to have led to a more correct view. But this seems not to have been the case. The theory of a contact ore-deposit, of a contact-vein between blue and brown limestone, remained the prevailing one.

The Spar fault-crevice has evidently served as the channel for the metallic solutions which formed the ore-bodies of the Spar mine. The ore as deposited follows evidently old water-courses in the limestone; the mineralizing solutions, ascending or descending in the Spar fault-fissure, attacked the more easily decomposing portions of the walls to a greater degree than other parts, and thence ate their way into the mass of the limestone, forming ore-chutes and chambers. Such a formation would naturally result in most instances in an ore-body without defined boundaries of the country rock. While the ore might take the place of an easily decomposing layer of the limestone in preference to other layers, the mineralizing solutions could hardly fail to attack and partly mineralize the adjoining layers of country-rock, as also the rocks adjoining their old channel.

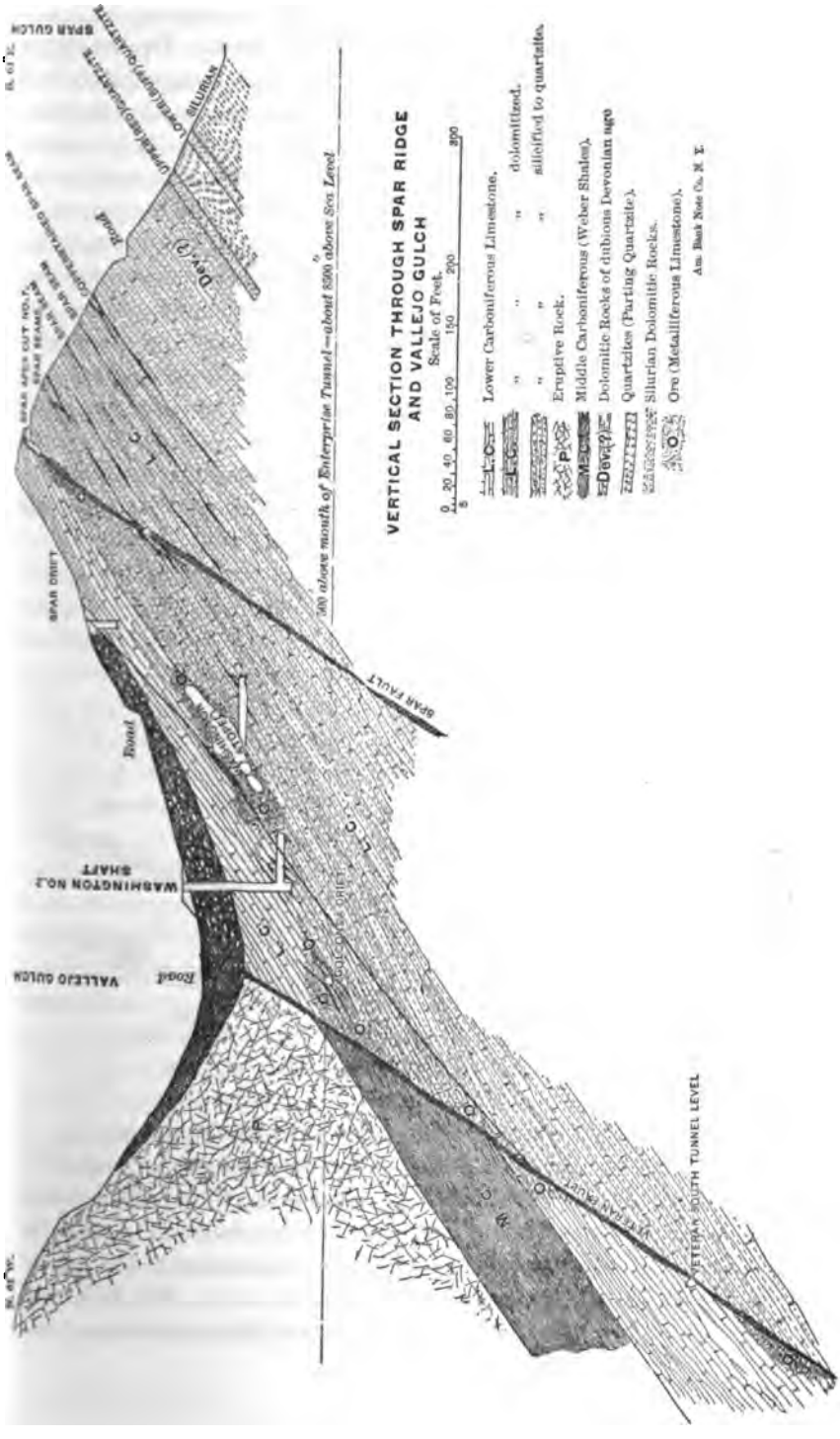
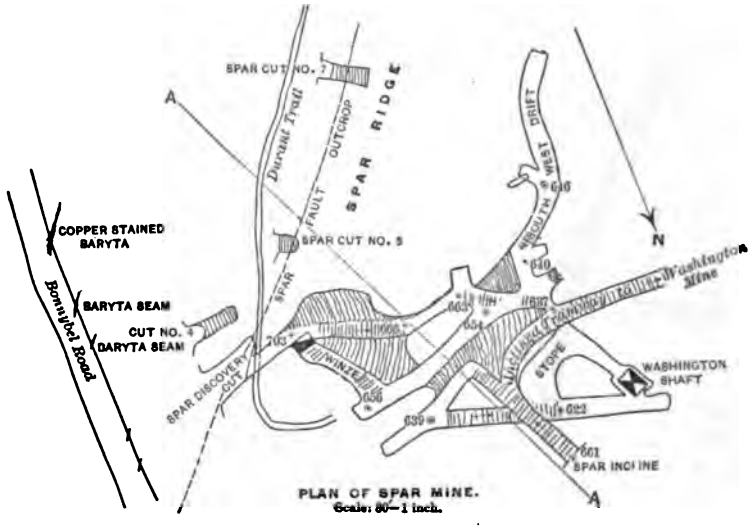

VERTICAL SECTION THROUGH SPAR RIDGE
AND VALLEJO GULCH
Scale of Feet.
0 20 40 60 80 100 150 200 300
Lower Carboniferous Limestone.
" " " dolomitized.
" " " silicified to quartzite.
Eruptive Rock.
Middle Carboniferous (Weber Shales).
Dolomitic Rocks of dubious Devonian age
Quartzites (Parting Quartzite).
Silurian Dolomitic Rocks.
Ore (Metalliferous Limestone).
Am. Bank Note Co. N. Y.
SPAR GULCH
Upper (red) Quartzite
Lower (grey) Quartzite
Summit
Spar Apex Cut No. 1
Spar Seam
Spar Drift
Tunnel
Dev.(?)
Spar Fault
Vallejo Gulch
Road
Washington No. 2 Shaft
Veteran South Tunnel Level

FIG. 6.

In the Spar mine, as far as opened, there is certainly no defined boundary anywhere between ore and country-rock. The one merges gradually into the other. The rich ore, with comparatively little lime in it, changes gradually to a lower grade ore, *i.e.* to a limestone less impregnated with lead- and silver-ore, and finally to a barren limestone. The latter change, however, has been exposed in this mine in few places, and then by accident; since only the pay-ore, or material out of which pay-ore could be got by hand-sorting, has been mined, while the poorer ore has been left behind in the mine, or thrown on the waste dump. The latter is now shipped to Leadville, as smelting-ore, only the large pieces of very low-grade limestone and heavy spar being thrown out. The floor, roof and sides of most openings in this mine show low-grade limestone and spar ore.

The richest ore-body in this mine was found in the dolomite. Here a coating of malachite covered and cemented the decomposed

FIG. 7.



dolomite pieces. This malachite was evidently the result of the decomposition of an original double sulphuret of copper and silver. This copper-stain on the brown limestone is therefore justly regarded by Aspen miners as a sure indication of the richness in silver of the limestone it covers. It is one of the few indications in this region, by which to distinguish ore from barren rock. But it is only present when the ore is unusually rich, and then not always. Be-

sides, there is another "copper-stain," resulting from the decomposition of copper-pyrites, which frequently disappoints the prospector, only a few ounces, or probably only a trace of silver being found by assay, when hundreds of ounces to the ton had been expected. This latter copper-stain is encountered quite often in baryta, and is one of the many reasons for the assumption that the formation of baryta in the limestone occurred at a different time and independent of the main argentic mineralization. Such copper-stain, of no value, is found, for instance, in a baryta seam, following bedding- and joint-planes in the dolomite, and coming to the surface on the upper side of the Bonnybel road. I have marked its location on the map of the Spar mine. This spar, although decidedly copper-stained, assayed, even in picked specimens, only 4 ounces silver per ton.

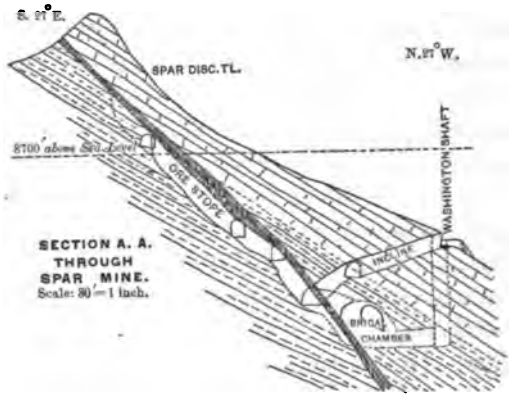
The Spar fault has been followed along its outcrop by several open cuts, small tunnels and inclines. Its contents of crushed material, which can be very plainly seen in these cuts, probably gave rise to the theory of a zone of crushed dolomite,—"like as if it had passed through a Blake Crusher," as one of the witnesses for the "apex" side in the Durant-Emma trial put it—which formed the gangue of the alleged contact-vein, and which was said to have resulted from the sliding movement of the Carboniferous blue limestone on the Silurian dolomite. In this we have an instance of a wrong application of a correctly observed fact. The crushed material is present, and the sliding movement is present—along the fault. Only the sliding movement did not occur on a division between geological formations, not even along the contact between brown and blue limestone. On the contrary, the movement occurred along the fault-fissure running nearly parallel with the strike of the strata, but cutting obliquely across the strata in the line of their dip, and thus, after the movement had taken place, bringing parts of the blue and brown limestone into contact, which before the movement had been away from the contact in the solid blue and brown limestone.

The dip of the dolomite strata can be measured very easily in Spar cut No. 4, directly south of the Spar discovery cut. (See Fig. 7.) As taken by the writer, it is N.W. 35° from the horizontal, while the dip of the Spar fault, as shown in the Spar mine openings, in the Spar discovery cut, and especially in Spar cut No. 7, is also N.W. but, on an average, 60° from the horizontal. (See Fig. 6.)

The claim of a contact-vein in the Spar mine, with a hanging-wall of blue limestone and a foot-wall of dolomite, is clearly not in accordance with the facts. If there be any fissure-vein, it can only

be the Spar fault-fissure. But the ore occurs mainly in the walls of this fissure, especially in the foot-wall, while the contents of the fissure itself are the least valuable parts of the ore-bodies. Whether other ore-bodies will be found at greater depth along the course of this fault is somewhat doubtful, as the lower layers of the dolomite are less easily decomposed than those in which we find the ore-bodies. It would, however, be well worth a trial, to see whether other ore-bodies exist along this fault at greater depth in the dolomite, taking the direction of the old ore-bodies, as shown by the

FIG. 8.



general direction of the stopes, as a guide in prospecting at greater depths.

The Washington Mine. (See Figs. 3 and 6.)—By following down the Spar inclined tramway the Washington mine is entered. Both mines now belong to the Aspen Mining and Smelting Company. A belt of limestone, either barren or only very moderately impregnated with silver-ore, intervenes between the Spar and Washington ore-bodies. The Washington mine was opened and worked, after the Spar mine had been partly opened, and after the theory of a contact-vein had taken firm root in the minds of Aspen miners. Nearly all development and work done in this mine has been, therefore, done along the contact (both the original and the fault-contact) of the overlying blue limestone and the underlying brown limestone. Any departure from this manner of exploring and working the mine which may have occurred was evidently forced on the management by the unmistakable trend of the ore-bodies, and was done reluctantly. A glance at the general map will show the large area in this mine occupied by stopes out of which rich ore has been taken. Some of

these stopes are 12 to 18 feet high, and contained practically nothing but ore, *i.e.*, decomposed dolomitic limestone sufficiently impregnated with lead- and silver-ore, to impart to it value enough to pay the cost of mining, shipping and treatment, and have some profit besides. And all the contents of these stopes have been such impregnated decomposed limestone, for that part which was not rich enough to pay in olden times has been shipped, or is now being shipped, from the huge waste-dump in front of the Spar incline, through which opening most of the hoisting from the Washington mine was done. There is plenty of low-grade ore left in the sides and the roof of these stopes; and, as I have been informed by men who were familiar with these mines while they were worked, much rich ore was left in the floor going down, and is now covered up by the waste. There is certainly a rich ore-body going down into the dolomite, near Washington No. 2 shaft (No. 37 on the general plan, Fig. 3); and I have no doubt that, in the close adherence of the management of the mine to the notion of a contact-vein, many opportunities were overlooked or disregarded, which an unbiased judgment might have followed down into the dolomite to other ore-bodies at a greater depth away from the contact.

In some parts of this mine, notably in the drifts and stopes running off from the Spar inclined tramway, a considerable amount of black carbonaceous shale occurs in the roof of the ore-bodies. This is either Weber shale, or it is the product of a decomposition of the crushing and grinding action of the movement of the limestone rocks on each other. In either case it would indicate a fault; and I have no doubt that such it is, and that this Washington ore-body, like the one in the Spar mine, has been formed in the neighborhood and along this fault-fissure, and that the original filling of this fault-fissure (in this instance the black clay shale) forms now the roof of this ore-body. In that case, however, there is a possibility of finding another ore-body on the upper side of this black shale. That this black shale is not the bottom of the regular Weber shale formation is evident from the fact that an upraise on the north side of the Spar inclined tramway passes through this black shale, which here is only about 18 inches thick, to the "blue" limestone (although in a somewhat decomposed condition) above it. This decomposed condition of the "blue" limestone above the shales augurs well for the existence of another ore-body. At this upraise the dolomite below the shales is not very richly impregnated with ore, and the decomposed "blue" limestone above assays a few ounces of silver

to the ton. This fault is probably the same as the one in the Spar mine, with only a locally flatter dip where the Washington ore body was deposited.

In the stope of the Washington mine, east of the Washington inclined tramway, leading down to Washington No. 2 shaft (No. 37 in Fig. 3), and running near the boundary-line of the Emma and Washington claims, is a clearly distinguishable fault, running a little east of north and west of south, with a nearly vertical position in some places, but varying very much in its dip at other points, and splitting up in places into several parallel fissures. It runs throughout the length of the stope, and it was only in its neighborhood on both sides that the rich ore was deposited which has been taken out of this mine. Where this fault crosses the Spar fault, *i.e.*, at the junction of the stope south of the Spar incline, and the stope east of the Washington tramway, an enormous body of rich ore was found, which made the Washington stope famous among Aspen miners.

Different drifts run east and west from the Washington tramway, following the contact as best they can—in most cases a difficult thing to do for any distance. As they get farther away from the north and south fault, the impregnation of the limestone with silver-ore decreases, until these drifts have reached the point where not enough silver is left in the rock to pay for extracting it.

Several winzes have been sunk into the brown limestone along this fault, and some rich ore has been taken from them. But the rock gets poorer at greater depth in the dolomite, as well as at greater height in the blue limestone. The ore has evidently been deposited along this fault-fissure, *i.e.* impregnating the limestone on both sides from the fissure. The impregnation and conversion of the limestone into ore are most decided next to the fissure, and the ore gradually changes into barren limestone, as the distance from the fault increases. But it is observable also, that this conversion of the limestone, next to the fault-fissure, is greatest at those places where pure limestone and dolomite have been brought together. The contact of rocks of different composition seems to be specially favorable, along such fault-fissures, to the conversion of both rocks into one.

The occurrence of fine slickensides in the ore-body has been cited as proof, that this fault occurred after the formation of the ore. But it only proves that subsequent movements of the rocks occurred along this fault, after the ore had been formed. There is not the least doubt that movements of the rocks still occur on Aspen moun-

tain. There is a large decomposition of the rocks going on inside the mountain all the time. Blue limestone is converted into dolomite, and dolomite into clay and fragments of decomposed rotten dolomite. The result of this decomposition is a shrinkage and settling of the rocks. That the movements of the rocks rather occur along open old fissures, instead of forming new ones, is also beyond doubt. I think that the fact of the concentration of the ore along the line of this fault is rather in favor of the priority of the fault to the deposition of at least the richer part of the ore, namely, the silver.

In the attempt to make connection between the ore-bodies of the Washington mine occurring near the "contact," and the "contact" itself in the Durant incline, another clearly-defined break in the formation was encountered, running in its course parallel with the strike of the strata, while cutting obliquely across them along their pitch. This last fault can be also observed in a beautifully smooth face of hard "blue" limestone, in the roof of the Durant incline (No. 26, Fig. 3) where it can clearly be seen that this fault does not follow the contact of the blue and brown limestone, but runs up into the blue limestone at an acute angle to the inclination of the strata.

The Vallejo Mine.—This mine is on ground in dispute between the Emma and Vallejo claims, both, however, now owned or controlled by the Aspen Mining and Smelting Company. The Vallejo shaft (No. 36, Fig. 3), although located in the bed of the Vallejo gulch, was only 90 feet deep when it reached the ore-body. The roof of the ore-bodies, which have been worked out in many places in this mine, is a mixture of decomposed porphyry and decomposed black Weber shales. The limestone in these mine-openings has been changed in many places, especially north and west of the shaft, into an almost pure quartz rock, which again was found richly impregnated with lead-ores or oxidized copper-ores, and contained much silver. A stope about 30 feet high (and more at some points) has been excavated mainly in this quartz-rock, and much good ore is still seen in its sides, roof and bottom. The change of the limestone into quartz-rock was found to continue in depth, as it was followed downward in the steep northwest incline to the lowest point reached by any of the mines in this part of the mountain, on the east side of the large Connamara fault. From the bottom of this incline a cross-cut extends northwest 40 feet, in the same quartz-rock. The same quartz-rock is found at the end of the southwest incline. From the large stope north of the shaft, the Golconda drift passes through blue limestone, impregnated with good ore all the way. The drift

connects these workings and the Emma mine with the Golconda shaft, which, in connection with the 400 feet inclined upraise to it from the Veteran tunnel, is used, as already remarked, for getting the ore from these mines down to the level and mouth of the Veteran tunnel and thence to the railroad track. Before reaching the Golconda shaft, the Golconda drift cuts the Weber shales.

I am not sufficiently acquainted with the formations exposed in the Golconda shaft and in the upraise from the Veteran tunnel to decide, whether the shales cut in the Golconda drift are in their regular geological position overlying the blue limestone, or whether, in passing from the blue limestone into the shales, the Golconda drift crosses the Connamara fault, or only a smaller fault parallel to the main fault. The whole of the thickness of the blue limestone cut by the Golconda drift, to its contact with the Weber shales, is impregnated with good lead- and silver-ore. Solid galena is also found in the shales and even in the tongues of porphyry which cut through the shales in this part of the mountain. Good ore is now mined along the contact of the Weber shales and the Lower Carboniferous limestone, near the Golconda shaft. The lower portion of the upraise from the Veteran tunnel followed an inclined zone of sandy, decomposed quartz-rock, with bunches and pockets of ore running through it. Whether this zone represents the filling of the large Connamara fault, or is only the silicified, decomposed rock at the side of the real fault, I cannot decide, having only once, and then hurriedly, examined the workings of the Veteran tunnel. This tunnel, near its breast, certainly cuts a big fault, and some of the drifts and inclines from the tunnel follow the line of fault. Rich ore has been struck next to a tongue of porphyry, and underlying the Weber shales, in an incline following down the fault from one of the drifts in the level of the tunnel.

The nature of the roof in a large part of the Vallejo mine, consisting of a mixture of porphyry and shales, changing at a lower depth to a roof of blue limestone, which, however, is very frequently also impregnated with ore, as is also the shale-porphyry mixture in places; the highly silicified condition of much of the Lower Carboniferous rocks in this mine; and the insignificant thickness of the blue limestone cut through in the Golconda drift (not more than 60 feet, even including some of the quartz-rock overlying the dolomite, found in places in the bottom of the big stope—horizontal measure in the line of the pitch of the strata)—make it certain that one fault, and probably several parallel faults, cross the ore-bodies of the

Vallejo mine and the Golconda drift. The rocks in this mine and in the neighboring workings of the Emma mine are so thoroughly decomposed (frequently to a crumbling, sandy material, more or less impregnated with ore), that it is often impossible to say, whether a rock was blue limestone or dolomite, before it was thus changed. The alteration of the limestone to quartz-rock, I have had occasion to witness in many different places in the mines on Aspen mountain. It occurs in the neighborhood of the large Connamara fault; in the workings of the Veteran tunnel, notably in the lower part of the uprise to the Golconda shaft (the upper part being unknown to me); in the Vallejo big stope and northwest incline; in the lower part of the Visino incline—always in places which, from their position, must of necessity be close to this fault (though it may not have been reached yet by the workings) in the neighborhood of the large downthrow of the Lower Carboniferous formation on the west, as shown to exist by the openings of the Connamara mine.

On the other hand, at some places in these mines, the porphyry also appears altered to a honeycombed or sandy quartz; so that it may be doubtful in some cases whether we have to deal, in such quartz-rock, with an original limestone or an original porphyry. Especially is this true, where we find such sandy quartz-rock impregnated with much ore, at the contact of porphyry and limestone, both occurring in less decomposed condition on the respective sides of this sandy zone.

The Emma Mine.—The Emma shaft (No. 35, Fig. 3) is located on the northwest slope of the Spar ridge. At 100 to 110 feet, it penetrated the "blue" limestone, and struck the upper edge of that large and rich ore-body, in which the big stopes of this mine and of the Aspen mine have been excavated. The extent and richness of this body, more than anything else, started the Aspen "boom" and gave rise to the "apex-side-line" litigation, only lately compromised. It was for the possession of this ore-body, or of its richest part in the Aspen ground, that the Durant party commenced the Durant and Visino inclines in support of their theory of a continuous contact-vein between blue limestone as a hanging-wall and brown limestone as a foot-wall.

In the Emma mine this ore-body is mainly in the dolomite underlying the blue limestone. The eastern boundary of the body follows the line of a fault, exposed in the No. 6 drift and upraise of the Emma mine, which connects with the Durant incline. Near the Aspen line a cross-cut, 112 feet long, has been driven into the

dolomite. Where the fault exposed in Emma No. 6 drift, at the east end of the ore-body, crosses this dolomite cross-cut, 55 feet from the mouth of the cut and about 62 feet horizontally from the "contact," about 18 inches of good ore was encountered, following this fault-fissure in its strike and dip. Other streaks and seams of good ore have been found along this drift in the dolomite, and the face of the drift 112 feet from its mouth, shows mineralized dolomite, assaying as high as 70 and 100 ounces of silver per ton. I cite these ore-seams, following the fault-fissures and their parallel seams and breaks in the dolomite, as a confirmation of the proposition, that the ore-depositing solutions followed these fault-fissures, attacking the limestone and depositing their metallic contents, wherever the limestone was most easily decomposed, and at the same time impregnated the sides of the fissures with ore to a larger or smaller degree. We find in the Emma mine, as in all others in this part of Aspen mountain, that there are no definable boundaries to the ore-bodies, separating them from the barren country-rock. The impregnation and replacement of the limestone with ore proceeded evidently from some point, line or plane, which, in the mines reviewed so far, we have seen to be mostly a fault-plane or fissure, and extended from this center, in an irregular manner, into the adjoining limestone, leaving harder, less easily changed portions of the limestone as "boulders" and "horses" of barren limestone or low-grade ore, surrounded by richer ore, while on the other hand good ore in irregular seams, chimneys, and pockets extended even into those portions of the limestone mass more removed from the centre of impregnation. That this impregnation and partial replacement took place, in this part of the mountain, mostly in the vicinity of the dividing line between the main bodies of Lower Carboniferous dolomite and blue limestone, is beyond dispute.

Much good ore has been left in this mine, in both roof and floor of the large stope. The mine was formerly much worked under leases, with the usual ruinous consequences. The lessees had, of course, no interest whatever in a continuance of the mine as a steady producer, and did not care to follow smaller seams, even of rich ore, as long as they could work in the main body. This old stope is, in parts, 18 and 20 feet high. Good ore can be seen almost anywhere in it, going up into the blue limestone roof (which shows very clearly the fissures, through which the ore-bearing solutions passed), and in the floor which, in a portion of this stope, is a layer of cherty dolomite, the old "footwall" but only six to eight inches thick, with very

rich ore underneath it. The lower levels of the old Emma mine were carried down the northwest of the big stope on what was evidently the continuation of this so-called foot-wall of dolomite. Good lead-ore was found here, and partly stoped out, mostly in the sandy decomposed blue lime. The lead-ore, in one part of the mine, in the old No. 14 drift of the Emma, occurred in two layers of nearly solid ore, separated by six to eight feet of barren, less decomposed blue limestone. But most of this ore, in this part of the mine, was of too low grade in silver to pay at that time. When the owners of the Aspen mine, during the winter of 1887–8, followed their blue limestone ore in the adjoining part of the Aspen mine, with a stope, keeping on top of the dolomite, they ran this stope under what was considered a part of the Emma dolomite foot-wall, showing that, in that portion of the mountain at least, the contact between blue and brown lime was not continuous, but that another fault, nearly parallel to the strike of the strata, passed through there. And it is between these two fault-fissures, the one passing through Emma No. 6 drift, the other passing through the Aspen shaft (No. 33, Fig. 3), about 210 feet from the surface, and followed down by the Aspen incline, that the main ore-body, hitherto worked in the Emma mine, is located. It is, however, certain that other ore-bodies exist northeast of the Aspen fault, and between it and the Connamara fault. In the Emma mine the two ore-bodies connect, or nearly connect along the line of the Aspen fault. The character of the ore, however, changes; and the ore which is now mined by the owners of the Aspen mine in that part of the mine west of the Aspen fault is very different from that taken from the big stope east of that fault. Much of the ore west of the fault is found in the blue limestone, and is heavy lead-ore, while most of the ore taken from the big stope east of the fault was copper-stained, brown limestone, carrying only a moderate amount of lead.

The Aspen Mine.—This mine is the best known, because the richest of all the mines in the mountain. The large quantities of enormously rich ore, exposed in the drifts of some parts of the mine, were, without doubt, the incentive to the protracted "apex-side-line" litigation. The Aspen shaft (No. 33, Fig. 3) in the bottom of Vallejo gulch, passes first through twenty feet of *débris* and loose porphyry, and enters the Weber shales, which it leaves at a depth of sixty feet. Thence it passes through 150 feet of hard blue limestone (Lower Carboniferous). A few feet above the bottom of the blue limestone it passes through a fissure which carries ore assaying about 80 oz. silver

per ton. This is the fault which I have mentioned in the review of the Emma mine, as passing through its lower workings. I have called it the Aspen fault, because all the lower workings of the Aspen mine are located either alongside or west of it. It has, in this part of the mountain, a course about N. 30° E., and dips N. 60° W. 60° from the horizontal. It cuts through the blue lime (which here dips about N. 30° W. at an angle between 35° and 40° on the east side of the fault, and of about 40° to 45° on the west side), until it reaches the first level of the Aspen mine and the western end of the old Aspen stope, which is only an extension of the old Emma stope. What has been said of the Emma stope is also applicable to this. Most of it was made when the mine was worked under lease. The upper or southeast end of this stope is connected with the end of the Durant incline by the so-called compromise drift or incline, an old inclined drift following the compromise boundary line between the Emma and Aspen claims, which overlapped each other. East of this compromise drift, the ore-body extends in the form of different layers of more impregnated limestone, separated by more barren layers of the same rock, to the west side of the fault passing through Emma No. 6 drift. This fault crosses the Durant incline a little below the long southwest drift, which runs from that incline across Emma and Aspen ground to the Connamara line (*a a*, Fig. 3). This drift follows this fault for a considerable distance, until the latter splits up, in the neighborhood of a N.W. and S.E. fault, crossing the drift, into several smaller fissures. This cross-fault is probably an extension of the Bonnybel fault, mentioned in the description of the surface-geology of Spar gulch. At the foot of the Durant incline, an opening has been made, thirty to forty feet high, following a steep, ore-bearing fissure, which seems to have been the main channel for the ore-bearing solutions forming the rich Aspen ore-body. The continuation of this fissure would carry us to the surface about at the point where the Hutchinson incline (No. 23, Fig. 3) has been sunk. This incline was sunk between limestone on the east and Weber shales on the west, following a fissure at an angle of sixty degrees. The fissure carries ore in small quantities, but assaying as much as 75 oz. silver per ton. This same fissure has been found at the end of the Emma cave adit (No. 30, Fig. 3), about forty feet below the bottom of the Hutchinson incline, and also in a short cross-cut in the floor of the Emma caves themselves. At both of these places, good ore, although not in large quantity, was found. The same fissure has been followed in highly decomposed limestone on the second

level south from the Emma Cave workings, where very rich ore has been found in it in bunches and seams. From the south end of this level a winze has been sunk along the pitch of the fault-fissure to a connection with the compromise drift. This winze showed more or less ore all along to its connection with the ore in the compromise drift.

This is the only fault-fissure on Aspen mountain which, to my knowledge, has ever been followed up to the surface. When we compare this fissure and its mineralization with the so-called "contact vein," exposed in the Durant incline, there can be no doubt which was the real channel of the ore-depositing solutions.

The ore-body in the Aspen mine, between this fault (which I have called the Hutchinson fault, after James Hutchinson, who was the first to trace this ore-bearing crevice through the Emma and Aspen workings, to locate it on the surface, and to start the incline, sunk on it from the surface down for over 40 feet) and the Aspen fault, is mainly in the dolomite. The impregnation, however, does not stop at the blue limestone, but in many places reaches up into it. But there is more and richer ore found, in that part of the mine, in the brown limestone than in the blue.

The richest place in this zone of the mine is in drift No. 2 (*b b*, Fig. 3), where a large body of ore, assaying thousands of ounces of silver per ton, is exposed on all sides of the drift for about 50 to 60 feet. This is the "show-drift" of the mine. Here the ore extends for a depth of 20 or 30 feet, and more, down into the brown lime, following mostly the smaller slips and cross-faults, which are visible everywhere through these workings. All the faults described, the Emma No. 6, Spar, Hutchinson, and Aspen faults, and one or two others yet to be mentioned, are parallel step-faults, running about N. 30° E., situated at short distances from each other, and seemingly getting closer to each other as they approach the region of the large downthrow of the Lower Carboniferous formation, exposed in the Connamara workings further west. Whether this Connamara fault is really one large fault, or a series of step-faults close together, has not been satisfactorily developed yet in the existing mine-openings.

The Aspen mine has several cross-cuts into the blue lime, some of them extending 100 feet to the west of the Aspen fault. All of these show bodies of good ore in any of the breaks crossed, parallel to the fault. The latter has been followed down on its dip by the old Aspen incline, from the old bottom of the shaft, 220 feet below the surface, to an additional depth of about 100 feet. From this in-

cline the second level of the mine was run along the fault, and two cross-cuts, called the East and West cross-cut, were run northwesterly into the blue limestone hanging-wall of the fault. The east cross-cut struck a parallel fault about 30 feet from the Aspen fault, and the west cross-cut encountered the same at a distance of 45 feet, showing a convergence of both faults towards the northeast. It is in the drifts and stopes run along the Aspen fault, that the gradual change of blue limestone into hard dolomite, thence into softer dolomite, decomposed dolomite, and, finally, into the usual cave-filling of this region (consisting of a red ferruginous clay, decomposed rotten dolomite, breaking very easily into sharp-edged small pieces, and of boulders of undecomposed blue limestone, protected by the coating of clay from farther decomposition) can be best observed. The dolomitization here follows the fault, and forms irregular lenticular masses, mostly in the foot-wall of the fissure. In some places the fissure has been completely filled by solid galena, uniting firmly both its walls. In other parts of the mine, I have observed this complete solid filling of a former fissure by calcite, but only in northeast and southwest fissures, from which it may be inferred, that the tangential pressure in the earth's crust, causing the slipping of the rocks along the faults, has come to an end as regards this direction, while it probably still exists and movements of the rocks are still going on in other directions.

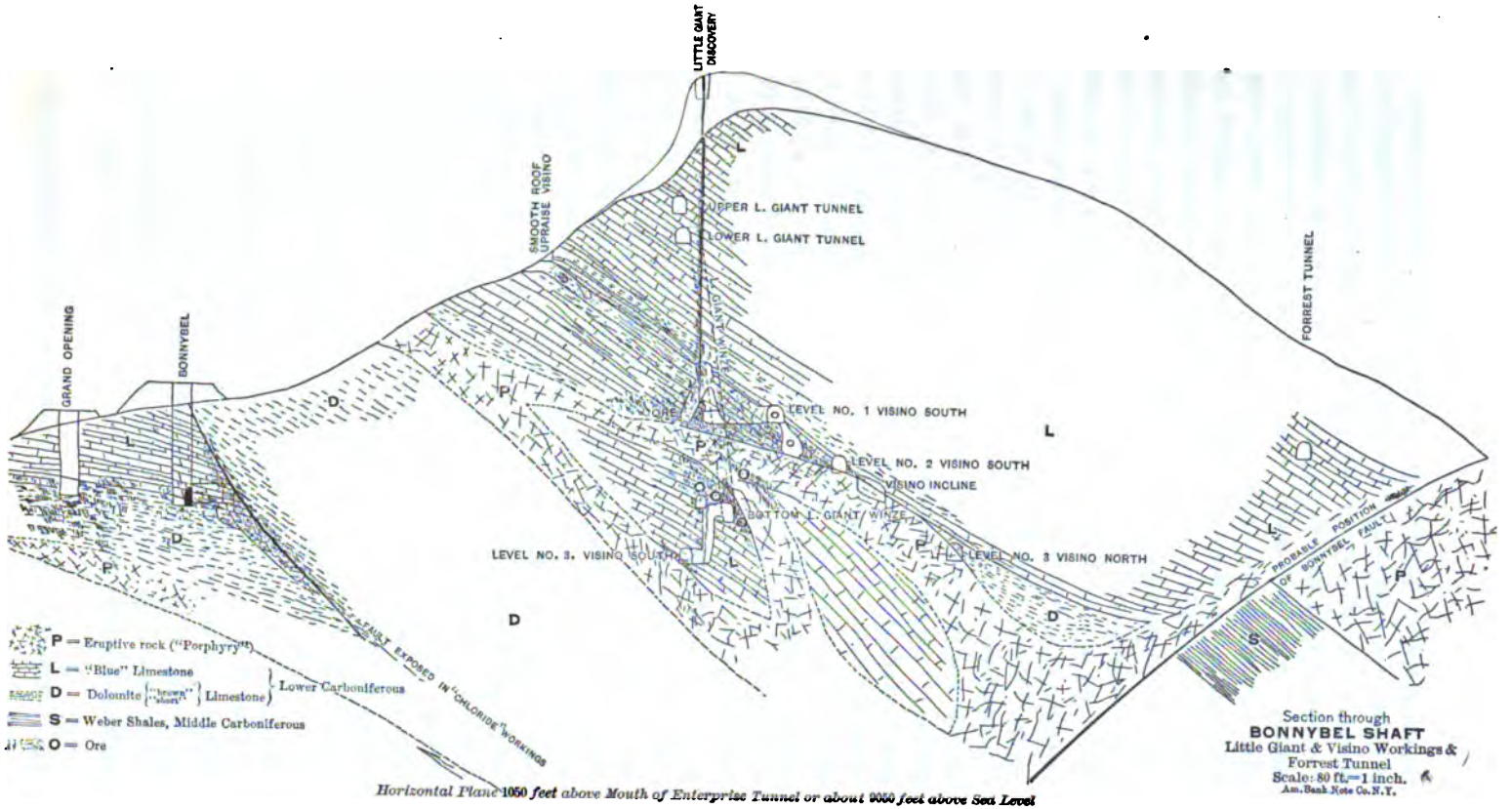
It is a pity, that at least one of these cross-cuts has not been driven farther out into the blue limestone and beyond it for some distance. At the end of the west cross-cut from the second level, 100 feet from that level (*c*, Fig. 3), gray clay shales with an abundance of disseminated iron pyrites cut off the blue limestone. An analysis of this rock gave about 60 per cent. silica, 20 per cent. alumina, and 15 per cent. iron pyrites. The question is, whether we have to deal here with a decomposed pyritiferous porphyry, or with some local modification of the Weber shales.

A further extension of these cross-cuts might not only increase our knowledge of the details of the geological structure of this part of the mountain, but might also lead to the discovery of good deposits of ore along the line of the Connamara fault. This suggestion holds good also for the Emma and Vallejo mines farther north. I have marked in the cross-section through the Washington No. 2 shaft (Fig. 6), the regions along that fault, where I think ore-bodies might be found in the Golconda claim. In all that part of the Aspen mine west of the Aspen fault, the ore-bodies are mostly in the blue

limestone. Frequently there are several such ore-bodies, separated by zones of comparatively barren rock. In their distribution and extent, these ore-bodies of the Aspen mine, west of the Aspen fault, show more clearly the influence of these fault-fissures on their origin and formation, than any others, so far as I know, on the mountain.

The Visino Mine.—This mine is mainly on the Durant claim; but its workings extend into several of the adjoining claims. At the mouth of the Visino tunnel an intrusive sheet or a dike of porphyry occurs between blue limestones. The blue limestone in the vicinity of the porphyry is changed to a large extent into marble, as is also some of the dolomite. The Visino tunnel starts in marbleized blue limestone and runs directly into solid porphyry, through which it penetrates again into marbleized blue and brown limestone, about 130 feet from its mouth. Here the first level of the mine runs off westward. At 150 feet from its mouth the tunnel is changed into an incline, which follows down practically on top of the porphyry sheet, penetrated in the tunnel, at an average dip of about 30° to 35°, to the third level. The dip of the top of the porphyry sheet, at first flatter than the incline, gets much steeper at level No. 3; and the porphyry disappears in the floor of the incline, a short distance below that level. This sheet is the common white porphyry of the region, containing very little iron pyrites. Below No. 3 level, and down to No. 6, occurs in the incline and in the beginning of the different south levels another sheet of pyritiferous porphyry, overlying the first-mentioned body of porphyry, and showing local inclusions of dolomitic limestone layers. The top of this sheet, at least where cut by the Visino incline (which, however, seems to follow, more or less closely, the summit of a local saddle in the porphyry), is still flatter than the top of the first sheet, but at No. 6 level it assumes a steep dip, and also disappears in the floor of the incline. The steep dip of these two porphyry sheets at No. 3 and No. 6 levels of the Visino incline correspond to the lines and the dips of two faults which cross the formations nearly on the line of the dip of the strata, are approximately parallel to the Bonnybel fault, and have a very variable dip. Between these two faults, a body of Weber shales, some of them very carbonaceous and black, has been penetrated by the incline. The direction of the fault, at the S. W. edge of these shales and dividing them from the overlying Lower Carboniferous limestones, and its pitch, are such, that the south side of incline follows the fault very closely between levels 4 and 6. This leaves only a thin shell

FIG. 9.


LITTLE GIANT DISCOVERY
SMOOTH ROOF UPRAISE VISINO
UPPER L. GIANT TUNNEL
LOWER L. GIANT TUNNEL
FORREST TUNNEL
GRAND OPENING
BONNYBEL
LEVEL NO. 1 VISINO SOUTH
LEVEL NO. 2 VISINO SOUTH
VISINO INCLINE
BOTTOM L. GIANT WINZE
LEVEL NO. 3. VISINO SOUTH
LEVEL NO. 3 VISINO NORTH
PROBABLE POSITION OF BONNYBEL FAULT
FAULT EXPOSED IN "CHLORIDE" WORKINGS
P = Eruptive rock ("Porphyry")
L = "Blue" Limestone
D = Dolomite {"brown" / "blue"} Limestone } Lower Carboniferous
S = Weber Shales, Middle Carboniferous
O = Ore
Horizontal Plane 1050 feet above Mouth of Enterprise Tunnel or about 9050 feet above Sea Level
Section through
BONNYBEL SHAFT
Little Giant & Visino Workings &
Forrest Tunnel
Scale: 80 ft. = 1 inch.
Am. Bank Note Co. N.Y.

NOTE.—This section passes nearly through Nos. 3, 6 and 19 on Fig. 3.

of black shales between the south side of the incline and the Lower Carboniferous rocks, a circumstance revealed by the caving-in of this thin shell of clay shales, after they had been exposed to the air for some length of time. This body of Weber shales is evidently the one which crops out below the limestone bluff traversed by the Forrest tunnel, and which has been cut on its other, the N. E. side, by the end of the Durant Apex tunnel No. 8. (See Fig. 3.) These faults, in conjunction with others and with the intrusive sheets, and the tongues of porphyry penetrating into the limestone in the neighborhood of the main bodies of these intrusive sheets, greatly complicate the structure shown in the workings of this mine.

The main ore-body in this mine and excavated in huge chambers or stopes is found mainly in the blue limestone, which is here marbleized and dolomitized to a large extent at the contact of the blue limestone and the porphyry. The regular contact between the overlying mass of blue limestone and the underlying dolomitized limestone, although quite flat in dip in a large part of the mine, has been found completely barren wherever encountered in this mine. The ore occurs in the blue limestone mainly on top of and (in the lower workings from the Little Giant winze) underneath the porphyry, forming large chambers and pockets in the decomposed blue limestone. (See section through Bonnybel Shaft, Little Giant and Visino workings and Forrest tunnel, Fig. 9.) One of the larger stopes occurs on level No. 4, where the limestone and probably also part of the porphyry next the limestone, has been changed into a mass of porous quartz-rock, crumbling easily and impregnated with ore. In levels 5 and 6 no ore is found, except along a crevice running north of west and south of east and cutting nearly vertically through the sedimentary rocks. Good rich ore is found in the immediate neighborhood of this crevice, while the contact of blue limestone and dolomite, as well as the rocks themselves, are barren. This vertical crevice can be easily traced throughout all the levels of the Visino mine, and passes right through the ore-stopes.

Another vertical seam, carrying ore throughout its length or depth, is that on which the Little Giant discovery cut is located, and along which the Little Giant winze was sunk from the lower Little Giant tunnel. (See Fig. 9.) This seam carries good ore, even where very narrow in the hard blue line, and it is easily traceable and very persistent in its extent. It is shown in the section through the Visino mine, where also the preponderating influence of the

porphyry on the formation of the ore-bodies of this mine is clearly discernible. That the porphyry has been the source of the ore-deposits of Aspen mountain is more than probable, but it is hardly as evident in any other case as in this mine.

The lower part of the Visino incline, from a point about 100 feet west of level No. 6, is run on a uniform grade of 45°. It cuts through some very large breaks and fissures and through some very hard ground. I think it affords a splendid proof, driven as it is on a uniform grade, without any regard to contact or character of rock, that wherever the Lower Carboniferous limestone is in the vicinity of any fissure affording a channel for the ore-bearing solutions, it is possible to get from a trace to a few ounces of silver per ton of limestone, *i. e.*, that the whole zone of Middle Carboniferous limestone has been slightly impregnated with silver-ores along these fissures. And not only is this true of the Lower Carboniferous rocks of the region, but also of the other rocks, though in a less degree.

At its lower end, the Visino incline penetrates a large body of highly silicified limestone, some of it also considerably impregnated with silver-ores, assaying 10 to 15 oz. silver per ton in spots, and after passing through this zone, encounters a much broken zone of dolomite, beyond which it passes into a zone of blue limestone traversed by large fissures filled with clay (probably the edge of the Connamara fault), which has also been reached by the southwest drift from the Durant incline, near the Connamara line. Here good ore was found along the break, between dolomite and blue limestone, the break between both rocks dipping very steeply to the east.

The Connamara Mine.—The Connamara (No. 31, Fig. 3) shaft was sunk on the slope of the porphyry ridge west of Vallejo gulch. It is only about 250 feet southwesterly from the Aspen shaft, and the shaft-collar is only a few feet more than a hundred feet higher than that of the Aspen shaft. But while the Aspen shaft reached the "contact" at a depth of 210 feet, the Connamara shaft has been sunk 620 feet, or about 300 feet below the horizon at which the Aspen shaft reached the "contact," and it is still in the blue limestone. As the line between these shafts is nearly that of the strike of the sedimentary rocks, there must be a folding or faulting of the strata between them, causing a large downthrow to the west. There is no sign of a folding of the strata in the west cross-cut drift from the second level of the Aspen; nor are there any indications of it in a drift run at a depth of 600 feet in the Connamara eastward towards the Aspen mine. But

a number of faults have been cut in the Aspen workings to the west, and at least two fault-fissures in the drift from the Connamara shaft. It looks, therefore, as if there were a series of faults, each with a downthrow of the west or hanging-wall.

While some ore has been found in each of these faults, some of them seem to have favored more than others the impregnation of their walls with ore. The block of ground between the Aspen fault and the next one west of it (diverging from it a little towards the south, at least locally, opposite the second level workings on Aspen ground) and the ground east of the fault passing about 70 feet east of the Connamara shaft at the depth of 600 feet, seem to be the most richly impregnated parts of this region of the mountain. Both these zones, at least down to a depth far below the 600-foot level of the Connamara, remain on Aspen ground for the whole length of that claim, and hence the developments from the Connamara shaft have so far served only to add considerable value to the Aspen claim. The richest ore in the Connamara workings has been encountered in running about 300 feet south of the shaft, from their side-line drift into Aspen ground, following a cross-fissure. About 50 feet east of the boundary line of the Aspen, and after apparently crossing what seems to be the same fault encountered in their east drift 72 feet east of the shaft (for these two fissures have the same dip and direction), the dolomite was found carrying true silver-ore (stephanite) plainly visible in the whole face of the drift, as impregnations and as seams in the rock. The same kind of ore was again struck about 40 feet from that face in a drift run south. Both exposures of ore belong evidently to one and the same rich ore-body. While they do not give any definite indication of its size, they promise a considerable body.

Some 400 feet vertically above this impregnation of dolomite with stephanite, cross-cuts from the southwest drift (*a a*, Fig. 3) run from the Durant incline have exposed in the Aspen ground a body of low-grade dolomite ore, about 75 feet wide, rather low in silver, but interspersed with richer streaks. There is no doubt that in this part of the mountain nearly the whole body of Lower Carboniferous limestone has been more or less impregnated with silver. This may prove to be a practical damage instead of an advantage, by scattering the silver in too much limestone, instead of concentrating it, as in the other parts of the mines. Still the discovery in this Connamara drift argues pretty well for finding rich bodies of ore in this part of the mountain. Numerous cross-faults and breaks have been

encountered in driving the Connamara drift, the first of which cuts off the fault-fissure followed by the drift. In that fissure was found a lot of "black talc"—quite a frequent filling of these fissures. It may be a product of decomposition of the Weber shales, leached into the fissures from above, or it may be a product of the crushing and subsequent decomposition of the limestone, colored by the organic matter, of which there is a considerable amount in some layers of the limestone. In any case, its presence indicates movements of the rocks on both sides of it; and it is mostly found in recognized fault-fissures.

The Bonnybel, Chloride and Silver Star Mines.—The Chloride mine (Nos. 2 and 3, Fig. 3), although not worked at present, enjoys the distinction of having been the first shipper of ore from Aspen. The first work done on this claim was on a pocket of very rich ore cropping out on the surface near the Silver Star line. Here a fault in the formation is clearly visible, and has been followed along its strike by a drift for about 30 feet. This drift, however, is now caved in. After exhausting the first pocket, two inclines were sunk farther north. In the one nearest to the original pocket good ore was found along the fault. That the connection of this fissure with the extent of the orebody was not recognized, is very evident from the shape of the workings. The same fault was again encountered in the North Chloride incline, and here a large body of rich ore was found alongside of it, near the contact of blue and brown limestone on the east side of the fault, but, in its largest and richest part, entirely in the dolomite on the west side of the fault. The regular contact of the blue and the brown limestone appears in No. 14 Durant Apex tunnel (No. 16, Fig. 3) over 150 feet west, and nearly 100 feet above the dolomite, which is, at that depth of 100 feet below the "contact," converted into rich ore. These Chloride workings connect with the old Grand Opening (No. 4, Fig. 3) and Bonnybel shafts (No. 5, Fig. 3), both of which are east of the fault at their bottoms. At the northern end of these Chloride workings on Durant ground, a winze was sunk 70 feet, following the smooth hanging-wall of the fault, but not of the ore; for this hanging-wall, and also the foot-wall, are in places strongly impregnated with silver-ore. The bottom of the winze is about 70 feet in the dolomite, below the contact on the west side of the fault, and on the hanging-wall side of the fault at least 170 feet or more below the contact of the dolomite with the overlying limestone. The latter contact, if not absolutely barren at this place, is certainly much less impregnated with

silver, and is therefore less valuable rock in place, than the dolomite 170 feet, geologically speaking, below it at the bottom of this winze.

This winze is connected with the south incline of the Bonnybel mine (No. 7, Fig. 3), the owners of which have lately taken up the theory of a continuous contact-vein between blue and brown limestone, and have been applying it against the originators of the theory, the Durant claim-owners. The Bonnybel is working an ore-body immediately below the Visino mine, cropping out on the steep slope of the mountain, south of the Bonnybel fault, the northwest and southeast course of which forms the northern end of this ore-body. The ore is a highly impregnated limestone, much marbleized, and is found in both kinds of limestones, but principally in the dolomite near the junction of the two. The limestone is traversed here in all directions (as above in the lower workings from the Little Giant winze) by porphyry tongues and sheets, which seem to follow the joint-planes of the limestone rather than the bedding-planes. They are probably branches of a larger porphyry dyke, which will probably be found at greater depth in the fault-plane running through the Silver Star workings, the Chloride workings, the south incline of the Bonnybel claim, and the winze in the old Chloride workings connecting with the south incline of the Bonnybel. In the bottom of the south incline of the Chloride mine a portion of this porphyry dyke, its upper end, as it were, has been encountered, and a winze has been sunk about 25 feet, following its contact with the underlying dolomite. That this fault has served as a channel for the ascending porphyry is evident from the tongues and sheets of porphyry encountered in all the Bonnybel workings in the neighborhood of the fault. The influence of these porphyry bodies in enriching the limestone in their neighborhood is quite evident in the Bonnybel workings, where the richest ore-bodies have always been found in the neighborhood of the porphyry tongues and sheets.

The Silver Star, after vainly trying by tunnels and inclines to find a contact-deposit of ore, has lately struck a good body along the fault-fissure in the dolomite, about 150 feet below its contact with the overlying blue limestone. The contact of the two kinds of limestone, passing frequently through a solid layer of limestone, crops out on the bluff about 150 to 170 feet above the discovery on the Silver Star claim. That contact is as barren of silver as any piece of limestone on Aspen mountain. Still, it has been claimed as a part of the continuous "contact-vein" of the Durant claim. The definitions of lodes and veins in the construction of the United States

statutes have been sometimes novel and wonderful; but it was reserved for this Aspen litigation to claim something as a vein or lode which in places is the indistinct division-line, following no seam at all, between the dolomitized and the unaltered parts of a solid layer of limestone.

V. Conclusions.

All the observed features of these mines, in their bearing on the character of the ore-deposits on Aspen mountain, warrant me in the conclusion that the ore is an altered limestone, even where it is now solid galena. For it is possible to trace in these ores all the different steps in the transition from limestone to solid galena. We find ore which is almost pure limestone with a slight impregnation of galena, then limestone heavily impregnated, then masses, in which bands of galena alternate with bands of limestone impregnated with galena, and so on, until we find the solid galena. But even this, when analyzed, generally reveals its origin by the presence of a few per cent. of carbonate of lime.

We find that the silver does not follow the lead in these ores, *i.e.*, that the silver contents of an ore bear no relation whatever to its contents in lead. Most of the richer silver-bearing ores are either true silver-ores, like polybasite and stephanite, or show a stain of carbonates of copper, resulting, probably, from the decomposition of bisulphides of copper and silver.

We find in certain portions of the mines much heavy spar, which, however, occurs not only in those parts of the limestones impregnated with ore, but very frequently in parts entirely barren, and also as well in the porphyry as in the limestone. Professor Lakes, in his report on the geology of Aspen, declares that analyses, made in the laboratory of the School of Mines at Golden, show the porphyry to contain an appreciable amount of oxide of barium. We may, therefore, consider the porphyry as the source of this mineral, which agrees very well with the location of the places where it is found most abundantly. Where the baryta occurs in the ore-bodies, as in the Spar mine, it contains very little silver-ore, and the larger pieces of it are sorted out of the ore as a detriment to its value, partly on account of its refractory behavior in smelting, but also, and mainly, because of its low grade in silver.

The conversion of the limestone into ore has taken place by metasomasis, *i.e.*, the gradual substitution of particles of ores for particles of limestone, thus changing the limestone partly, and in rare instances completely, into ore. This impregnation of the limestone

proceeded from fissures in the limestone and extended thence into the mass. We find the richest ore and the most altered limestone following the direction of these fissures, and we see the ore gradually passing into barren and only slightly impregnated limestone, as the distance from these fissures increases. These fissures are mainly fault-fissures, and we find the ore-bodies to be the largest and richest along the fissures where the faulting brought different kinds of rock in contact with each other: slate with blue limestone or with dolomite, or blue limestone with dolomite. Some of these fault-fissures have, evidently, been more favorable than others to the formation of ore-bodies along their line. These fault-fissures have also frequently served as channels for the eruptive rocks, as is shown by the porphyry tongues or dykes found in some of them.

While the contact of the blue limestone with the underlying dolomite has evidently been a favorable region for the formation of ore-bodies, such bodies have only been formed in that region along fissures cutting through the limestones and the contact, which fissures have evidently been the channels for the metallic solutions impregnating the limestones and forming the ore-bodies.

There is no doubt, while the juxtaposition of limestone and dolomite, especially that caused by the faulting of the formation, is a favorable region for the formation of ore-bodies, too little attention has been paid so far, in the prospecting and developing work of the mines, to the exploration of these fault-fissures away from the contact.

Wherever the contact has been found in an undisturbed condition, it has always been found barren, and often only as an imaginary division-line in a solid layer of limestone. And in most cases, where the contact has been found to contain ore, *i.e.*, where the rocks on both sides were found impregnated with ore, it has been found to be a fault-contact, and not one caused by the original deposition of the formation.

Finally, while some of the richest ore-bodies on Aspen mountain have been found removed from the mass of porphyry in the interior of the Lower Carboniferous limestone, we have seen, in the Visino, Chloride and Bonnybel mines, that intrusive sheets or dykes of porphyry in the Lower Carboniferous limestone, are, on Aspen mountain as elsewhere, very favorable localities for finding good bodies of ore, and that the contact between such sheets or dykes of porphyry and the limestone is well worth prospecting.

The ore-deposits of Aspen mountain occur throughout nearly the

whole of the thickness of the Lower Carboniferous formations, although the lowest dolomite beds, immediately overlying the parting quartzites, do not seem to have been very favorable to mineral deposition. A zone of the Lower Carboniferous limestone, from 40 or 50 feet below the top to 200 feet below the top, or about 150 feet above the parting quartzites, seems to have been specially favorable to mineral deposition; and since, on Aspen mountain, the boundary between the dolomitized and the unaltered Lower Carboniferous limestone lies (though with considerable local variation) within the limits mentioned, the ore-bodies may be said in a certain sense to be *near* the contact between "blue" and "brown" limestone. The ore-deposits of Aspen mountain certainly neither *follow* the contact, nor are they in any fair sense of the word a "contact-vein," least of all a continuous one, which only exists in the imagination of those who speak of the "great mother lode of Pitkin county." They are best described as impregnations of the Lower Carboniferous limestone with lead and silver ores, concentrated, along certain fissures, which have probably acted as channels for the metalliferous solutions, to irregular ore-bodies, passing gradually into less impregnated, and finally into the barren mass of the country-rock.

THE DEVELOPMENT AND STATISTICS OF THE ALABAMA COAL-FIELDS FOR 1887.

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(Birmingham Meeting, May, 1888.)

[The statistics contained in this paper were collected for the United States Geological Survey and communicated to the Institute, by permission, prior to their publication in the report on the Mineral Statistics of the United States for 1887.]

THE total coal-production of Alabama in 1887 was about 1,950,000 short tons, the spot-value of which, at the mines, was $2,535,000.

The coal-regions of the State lie in the extreme southern part of the Appalachian coal-field, the northern part of which lies along the western New York-Pennsylvania State line; between these limits, New York on the north and Alabama on the south, portions of the

field are contained in Pennsylvania, Ohio, Maryland, Virginia, West Virginia, Kentucky, Tennessee and Georgia. During the past year no extended surveys have been made which give any more exact knowledge as to the area of the Alabama fields, which, according to the latest estimates made by Mr. Henry McCalley of the State Survey, cover 8660 square miles.

Many local names are employed by local mining men to designate special parts of the region, although for general descriptive purposes a division into the Warrior, Cahaba and Coosa is quite sufficient—the first-named field, comprising all the area underlain by coal-measures and drained by both forks of the Warrior and their tributaries, and the Tennessee river and its tributaries in Alabama, being more than ten times as large as the area of the other two combined.

The Coosa field extends northeast and southwest from Calera to Eastport, a distance of about 60 miles, the width of the field ranging from 5 to 12 miles. It has been recently estimated that the Coosa field embraces about 30 square miles in the northwestern part of Calhoun County, about 150 square miles in Saint Clair County, and about 235 square miles in Shelby County, making an aggregate of 415 square miles. It has been also estimated that the Cahaba field includes about 50 square miles in Saint Clair County, 100 in Jefferson County, 160 in Shelby County and 125 in Bibb County, aggregating 435 square miles. Only 75 square miles in Bibb County are free from drift, so that the area depending upon the coal-measures for its soil is reduced to 385 square miles.

The relative importance of the several counties in the production of coal may be gathered from the following summary table:

Production by Counties, 1887.

County.	Production.	
	Tons.	Per cent.
Jefferson	1,384,000	70
Bibb	230,000	12
St. Clair	53,000	3
Walker	222,000	11.5
Tuscaloosa	9,000	.5
Shelby	52,000	3
Total	1,950,000	100

It has been estimated in a rough way that there are 108,394,000,000 tons of available coal in the Warrior field, in seams over 18 inches

thick, of which there are 50 seams included in the coal-measures which attain their maximum thickness of 3000 feet, more or less, near Tuscaloosa. Such estimates as these are of no practical value, since it is a matter of very little importance to the interests of the State or the coal-trade, whether the coal-fields contain 50 billion tons more or less, while there may be said to be in sight sufficient coal excluding all other fields to meet the wants of the entire United States for many years. These estimates, however, have been the basis for the most absurd statements as to the preponderance of coal in Alabama over that in other States, and it may be of interest in this connection to give some general figures for other coal-fields as a basis for comparison.

The thickness of the bituminous coal-measures proper in Pennsylvania is 2600 feet, to which must now be added 1200 feet—the thickness of the Pocono sandstone, No. X., which contains valuable coal-beds, which are now being extensively worked in the vicinity of Altoona,—making a total thickness of productive coal-measures of 3800 feet, containing workable coal-beds aggregating in thickness in many places 100 feet, more or less. In the Pennsylvania anthracite region the thickness of the coal-measures in the Pottsville basin is 3100 feet, containing workable coal-beds, having an average aggregate thickness of 120 feet, although in one part of the region the Mammoth coal-bed alone attains a thickness of over 100 feet, while a thickness to this bed of over 50 feet in extended areas is not infrequent. In the Pennsylvania bituminous region, there is estimated to exist, within easily attainable depths, in beds which are over 2 feet 6 inches thick, at least 33 billion tons of merchantable coal. In beds over 18 inches thick, and at all depths, there certainly exist in this region 120 billion tons of merchantable coal.

A very conservative estimate of the amount of merchantable coal in Illinois, in beds of present workable dimensions, is 30 billion tons. In Ohio and West Virginia the amount of merchantable coal contained is quite comparable with that contained in Illinois and Pennsylvania. While no practical advantage is to be gained by general statements to prove the importance of any one State coal-field, the above figures would hardly warrant the statement which is currently made, that Alabama contains a greater thickness of coal-measures and more available merchantable coal than any other State.

The geological structure of the coal-beds of Alabama is variable as to their dip. In the Warrior or more western field the coal-beds

lie more generally horizontal than in either the Cahaba or the Coosa field, where the coal-beds dip at all angles, in some cases being completely overturned.

The most valuable coal-bed developed in the State, and that from which the greater amount of coal is mined, is the Pratt bed in the Warrior field, although there are believed to be several other beds equally as good. According to Prof. E. A. Smith, State Geologist, the seam at Blocton in the Cahaba field (Bibb county) is over 6 feet thick, free from shale, and in every respect equal to the Pratt seam. There are other coals which coke equally well, though not mined or coked to the same extent as the Pratt coal. The thickness of the Pratt bed varies from 4 feet to 6 feet; the average thickness throughout the field for mining purposes may be considered to be 4 feet 6 inches. One of the most persistent geological features of this bed is the occurrence of a slate-parting, from 2 to 3 inches thick, 7 inches from the top of the bed. As the bed is mined, its product is generally 33 per cent. of lump-coal and 67 per cent. of nut and slack, the latter percentage being utilized mostly in the coke ovens. It is generally estimated that the Pratt bed will yield, on an average, about 4000 tons of merchantable coal to an acre. The New Castle seam, with an average thickness of 7 feet and over, with more or less slate as partings, called in places the "Big Vein," covering a much larger territory than the Pratt seam, is destined, it is believed, to be *the great coal-seam of Alabama.*

Statistics for 1887, from all the known mines in the State, were obtained through returns made by each company direct to myself, as special agent in charge of coal statistics for the United States Geological Survey, the names of a number of new coal-mining establishments being furnished by Prof. Eugene A. Smith, State Geologist. The total production for the year was 1,945,812 tons, including all the coal mined, whether used for making coke or directly shipped as coal.

If all the coal which has not been reported and which is mined at country banks for local consumption should be included, it is believed that 1,950,000 tons would be a maximum estimate for 1887. A summary of the returns received from the individual mines is given in the following table (p. 210).

The Pratt mines are operated by the Tennessee Coal, Iron and Railroad Company, which operates also the Tracy City and Whitwell mines in Tennessee.

The Pratt mines, one year ago, had a daily capacity of 2500 tons;

Name of Company.	Location.	Production.	No. of Mines.	Miners etc.	Wages.	Price at Mines.	Days worked
(Jefferson County).							
Tenn. C. I. & R. R. Co.	Pratt Mines.	711,606	4		$		
Henry-Ellen Coal Co.	Henry-Ellen.	*41,000	2				
Sloss Iron & Steel Co.	Coalburg & Brookside.	189,555	4				
Watts Coal & Iron Co.	Warrior.	23,000	3	75	.70	$1.20	200
Warrior C. & C. Co.	"	42,050	7	100	.70	1.20	250
Mabel Mining Co.	"	30,000	1	90		1.20	200
Pierce-Warrior Coal Co.	"	51,900	3	55 to 90	.70	1.20	
Hoene Consolidated Coal Co.	"	58,220	3	200	.70	1.20	252
Milner Coal & R. R. Co.	New Castle.	65,000	2				
Woodward Iron Co.	Woodward.	60,676	1	100	.50		310
J. S. Carr & Co.	Morrow's Mines.	110,000	1				
Total.....		1,383,007	31				
(Bibb County).							
Brierfield Coal & Coke Co.	Brierfield.	72,000	2	200	.70	1.50	250
Cahaba Coal Mining Co.	Blocton.	157,821	6	550	.55		250
Total.....		229,821	8	750			250
(St. Clair County).							
Coosa Coal & Coke Co.	Broken Arrow.	none	2				
St. Clair Coal Co.	Ragland.	52,141	1	50	.40	1.15	275
Total.....		52,141	3				275
(Walker County).							
Wolf Creek Coal Co.	Corona.	530	1	24	.72½	1.50	19
Penn Mobile Coal Co.	"	*44,000	1				
O'Brien Coal Co.	"	10,500	1	45	.72½	1.30	175
Va. & Ala. M. & M. Co.	Patton & Coal Valley.	60,000	3	265	.72½	1.37½	146
T. H. Dunn & Co.	York.	*18,750	1				
Frief & Moore.	Horse Creek Mines.	15,000	1				
Kansas City C. & C. Co.	Carbon Hill.	2,000	1	75	.55	1.40	24
B. M. Long & Co.	Cordova.	10,000	1				
E. Donaldson & Co.	Coal Valley.	14,000	2	50	.55-.60	1.37½	146
Black Diamond Coal Co.	Patton.	12,500	1	50		1.25	150
Norwell & Co.	Day's Gap.	13,758	1	26	.60-72½	1.25	100
Corona Coal & Coke Co.	Corona.	20,000	1	125	.62½ to .72½		
Total.....		221,088	15				
(Tuscaloosa County).							
A. Durie, and others.	Tuscaloosa.	5,250	1				
Alabama Insane Hospital.	"	3,000	1	4	.87½		310
Total.....		8,250	2				
(Shelby County).							
Montevallo Coal & Iron Co.	Aldrich.	36,555	1	75	1.00	2.50	260
Unreported.....		15,000					
Total.....		51,555					
Grand Total.....		1,945,812					

* No return received. Production estimated to be about the same as that reported for 1886.

at the end of 1887 this had been increased to 3000 tons, and it is planned now to increase this to 4000 tons during the early part of 1888, and to 5000 tons by the spring of 1889.

Although the proportion of the Company's coal mined in Alabama, which is coked, is not reported, a comparative idea may be gained from the following complete returns:

Statistics of the Tennessee Coal, Iron and Railroad Company, 1887.

Pratt mines production,	711,606 Tons.
Tracy City,	415,485 "
Whitwell,	12,529 "
Total production,	1,139,620 "
Total consumed at mines, including amount made into coke,	509,288 Tons.
Total shipped to outside markets,	630,332 "
Total production,	1,139,620 "

Outside of the above company, the amount of coal coked, as reported by the different companies, was as follows:

Sloss Iron and Steel Company,	63,600 Tons.
Watts Coal and Iron Company,	2,000 "
Warrior Coal and Coke Company,	2,042 "
Woodward Iron Company,	60,676 "
Brierfield Coal and Coke Company,	48,000 "
Saint Clair Coal Company,	15,047 "

An analysis of the coal produced by the Sloss Iron and Steel Company, made by Prof. J. L. Campbell and reported by the Company, as showing the average quality of the coal produced by their mines, is as follows:

Ash,	3.014
Fixed carbon,	65.075
Volatile matter,	30.745
Sulphur,	1.203
Water,	.935

The specific gravity of this coal is 1.295.

The Sloss Company is operating four drift-openings in the Pratt seam a mile back from Coalburg railroad station; two small openings have also been made near the depot. The coal-bed in this part of the field will average about 3 feet in thickness. A new mine has also been opened at Brookside station, 6 miles west of Coalburg,

from which it is expected good coal can be mined; the bed shows a somewhat greater thickness than at Coalburg.

The 95 bee-hive coke ovens, near the No. 4 and B mines, are kept constantly in blast, supplying coke for the Williamson furnaces in Birmingham.

The Sloss Iron and Steel Company is preparing to make extensive improvements about its coal plants. It expects in the near future to furnish the coal from these works for the 250 coke ovens at the two large furnaces in Birmingham, and also to make coke for supplying two new furnaces at North Birmingham. At No. 3 mine the tail-rope haulage-system has been introduced, which enables the coal to be moved more rapidly. The rope is 3300 feet long. At this tipple a Thomas coal-crusher and washer is in operation, through which the screenings from several mines will be run, adding to the quality of coke produced.

A plant of 63 of the Thomas coke ovens is nearly finished. This is a new oven for which great advantages are claimed over the bee-hive. It is built similar to the bee-hive, except that it is long instead of round. Each oven is 6 feet wide, 36 feet long and 5 feet high in the clear. A 14-inch wall separates each oven from the next. They are charged like a bee-hive from the top, taking a charge of 12 tons of coal to each oven. Steam-machinery, mounted on wheels, is run in front of the ovens, with which the charge is drawn out of the oven after the coke is made. The mouth of the oven is a few inches wider than the back part, allowing the charge to slide out freely. It is claimed that the coke can be economically taken from the oven and loaded directly into cars with this machinery. The ovens can be immediately recharged, thus keeping them at a greater heat and making a better quality of coke. These will be the first of this class of ovens that have been built in this section of the South. A number of new houses have been built by the Sloss Iron and Steel Company during the year. At present, work is a little slack in the mines. About 275 free laborers and 275 convicts are employed. The miner receives 60 cents per ton for run of mine coal.

The Watts Coal and Iron Company's mines are situated on the north bank of Warrior River, about 2½ miles southeast of the town of Warrior, in township 14 S. range 3 W. Owing to the burning of the machinery the mines were closed during the months of November and December, operations being resumed in January, 1888. In consequence, the total production for the year was much

smaller than it would otherwise have been. The coal-bed mined averages about 3 feet 6 inches thick. The analysis of the coal made by Mr. N. T. Lupton and of the coke made by Mr. Brainerd are reported by the company as follows:

	Coal.	Coke.
Ash,	1.92	4.780
Fixed carbon,	63.03	92 304
Volatile matter,	33.88	1.600
Sulphur,	1.20	1.116
Water,	1.17	.200

The Company is opening a new mine in the Pratt coal-bed on Blue Creek, southwest of Birmingham. The mines during 1888 will be prepared to produce 300 tons of coal daily. The coal from this mine, other than that which is coked, is shipped to the Louisville and Nashville railroad.

The Warrior Coal and Coke Company has seven drift-openings, which are all connected together as one mine, located near Coaldale; the mines were worked about 250 days during 1887. The average thickness of the bed, which is at present worked, is 3 feet; it is reported that there are three other coal-beds which underlie the entire area of the company's property, one 4 feet thick and the other, 2 to 3 feet thick respectively. An analysis of the coal which is at present mined, made by Mr. N. T. Lupton and reported by the company, shows:

Ash,	1.27
Fixed carbon,	65.12
Volatile matter,	32.24
Sulphur,	.56

The coal is shipped to local points in Alabama, to southwest Georgia, to Florida and to New Orleans.

The Mabel Mining Company's mines are located 1 mile north of the town of Warrior. The coal-bed which is mined has an average thickness of 2 feet, 10 inches; the coal is particularly adapted for steam purposes, and shows on analysis:

Ash,	1.27
Fixed carbon,	65.13
Volatile matter,	32.24
Sulphur,	.50

The mines employ at times from 75 to 100 miners, and the product is shipped to points along the Louisville and Nashville railroad, lying between Nashville and New Orleans.

The Hoene Consolidated Coal and Iron Company's mines, with the production of each for 1887, are as follows:

Name of Mines.	Location.	Production.
Alabama,	At Warrior,	26,324 Tons.
Jefferson,	3 miles S. of Warrior,	20,396 "
Brake,	2 " " "	15,000 (?) "

The Pierce-Warrior Coal Company is operating three mines in the vicinity of Warrior, Jefferson county, located as follows: Old Shaft mine, half a mile north of Warrior station; New Shaft, the largest and best mine, half a mile south of the station, and the Wolf Den Hollow mine. The New Shaft mine was not worked during 1887 alone. The present Company is a reorganization of the old Pierce-Warrior Coal Company, and took charge of the mines on March 1st, 1887. During January and February, 1887, 5500 tons had been mined by the old company; since March 1st, the new organization mined 45,400 tons, making a total for the year of 50,900 tons, 1000 tons of which were consumed at the mines. The coal-bed varies from 2 feet to 3 feet 8 inches thick; the coal is used principally for steam and blacksmithing purposes. The following partial analysis of the coal is reported by the Company:

Ash,	1.27
Fixed carbon,	65.18
Volatile matter,	32.24

The number of men employed at these mines varies from 55 to 90; miners are paid 70 cents per ton for mining; laborers earn from $1.00 to $2.00 per day. The coal is shipped to Nashville, Tennessee, and the cities of Decatur and Mobile, Alabama; the average selling price per ton during 1887, being $1.20 on cars at the mine.

The average thickness of the bed at the Alabama mines is 2 feet, 10 inches, while at the Jefferson and Brake mines, it is 3 feet, 6 inches. The coal is particularly adapted for steam and gas purposes. The mines were worked 252 days during the year, and there were employed, on an average, 200 men, both inside and outside. About two-thirds of the product of the mines were shipped to the Louis-

ville and Nashville railroad, and one-third to Montgomery, Alabama, Mobile, New Orleans, Selma and to several points in Georgia.

The two mines of the Milner Coal and Railroad Company are located in the vicinity of New Castle, in Jefferson County. Two coal-beds are worked at these mines, the Black Creek bed, from which 60,000 tons of coal were mined in 1887, and the New Castle bed, from which 5000 tons of coal were mined in the same year. This coal was shipped to Montgomery, Mobile and New Orleans, 60,000 tons being sent to market and the remaining 5000 tons consumed locally at the mines. The coal commanded, for run of mine coal, a price of $1.25 per ton at the mines.

The Woodward Iron Company is operating mines at Woodward, in Jefferson County; its coal-bed is opened by slopes. During 1887 the production of coal was 60,676 tons, all of which was consumed by the company in making coke for use in its two furnaces at Woodward. A new slope has recently been driven in the coal-bed, and gives promise of an extensive mine. The coal-bed at Woodward is 4 feet 8 inches thick, the coal being particularly adapted for coking purposes. The mines were operated steadily throughout the year, having worked 310 days; 100 men were employed.

The mines of the Brierfield Coal and Coke Company are located in Bibb County, near the edge of Shelby County, and 3 miles from Brierfield, with which they are connected by a standard-gauge railroad. During 1887 this company produced 72,000 tons of coal, of which 48,000 tons were manufactured into coke, and the remainder shipped to various points in Alabama, Georgia, Mississippi and Louisiana. Four coal-beds, each about 4 feet thick, are worked in the two mines of this company; their extent is practically unlimited. The coal is used for domestic purposes and for making coke. During 1887 the company employed about 200 persons, the mines being worked 250 days during the year.

The Cahaba Coal Mining Company's mines are located at Blocton, in Bibb County, 10 miles from Woodstock station, on the Alabama Great Southern railroad, and 29 miles south of Birmingham. The village of Blocton, where all its works are located, contains 350 neat houses, built by the company. The company also built and operates a standard-gauge road from Blocton to Woodstock, where a junction is formed with the Alabama Great Southern road. A junction has also just been made at Woodstock with the new mineral branch of the Louisville and Nashville railroad, giving two good

outlets. It is possible another branch from the East Tennessee, Virginia and Georgia road will tap the Company's line the present year, giving a short route to Anniston, where 2 large blast-furnaces will be supplied with coke by the Company. Up to 1887 the Company had but one mine; 5 new mines, from which shipments were commenced in November, were opened during the year. Two separate coal-beds are mined, locally known as No. 1 and No. 2. Two slopes and a shaft are driven into bed No. 1, and two slopes and a drift into bed No. 2.

The largest and most extensively operated mine of the six is the No. 2 slope, opening into the No. 2 seam. This opening was made into 6 feet of fine coal 1 mile back from the Company's headquarters. This is a double-track opening, following the coal down 1500 feet, at an average pitch of 9 degrees. Seven lifts have been opened. These lifts are 200 feet apart. The entries of each lift are run east and west from the slope-track, some of them extending back 1 mile from the slope. Mr. Thomas Belch has charge of the underground work, and about 700 tons of coal are hoisted daily.

No. 3 slope was recently opened in the No. 2 seam, 2 miles from No. 2 slope. This is also a double-track opening, from which a large amount of coal will soon be hoisted. Already 2 headings have been turned off from the first lift, and rooms worked in them, and two others are being turned off on the second lift. The slope-headings are being driven down as fast as possible, running double shifts. No. 6 is a drift opened into the same seam opposite the tipple of No. 3 slope. These two mines are in charge of Mr. Herbert Thomas. An additional force of miners will be placed in these mines as fast as room is made. In this seam the miners are paid 55 cents a ton for forked coal and 50 cents for run of mine coal.

No. 1 slope is opened in the No. 2 seam. This slope was idle last year, but has recently been repaired. The coal-bed at this opening has a dip of about 5 degrees. The seam averages 3 feet in thickness. A new engine, boiler and other machinery have recently been set up, and preparations made for hoisting a large amount of coal. Mr. James Wilcox has charge of the underground work. The miners working in this seam receive 80 cents per ton for run of mine. This slope is down 800 feet.

No. 4, another new slope opened into No. 2 seam, has been made 1 mile north of the Company's headquarters. It is now down 700 feet, going towards No. 1. Mr. James Waugh has charge.

No. 5 is a shaft sunk to No. 2 seam, midway between No. 1 and

No. 2 slopes, both of which are being driven towards it. When underground connections are made, this shaft will drain the other two mines. This will be an important work. Good machinery is being placed in position for hoisting coal and pumping the water that may accumulate in all three mines. The shaft measures 10 by 18 feet, and will be fitted up with an automatic hoist, built by the Crane Elevator Company. The mine will be in charge of Mr. Andrew Bryson. The same power that operates the shaft will draw water from the creek near by, and force it up into a large tank on the hill-side, 50 feet above the coke-ovens. The pump for this purpose has a capacity of 5000 gallons per minute, guaranteeing abundance of water for the coke-plant.

The following analyses show the composition of the coal from each bed; No. 1 having been analyzed by J. B. Porter & Co., of Cincinnati, and No. 2 by Mr. E. Nichols:

	No. 1.	No. 2.
Ash,	3.20	2.41
Fixed carbon,	55.76	60.75
Volatile matter,	41.04	34.12
	100.00	
Sulphur,	1.01	.48
Water,		2.24
		100.00

The specific gravity of No. 1 coal is 1.27.

The thickness of No. 1 bed is 3 feet 6 inches, and of No 2 bed, 6 feet. The character of the coal is hard bituminous, and it is particularly adapted for steam, gas and domestic uses.

The product of the mines is shipped to local points in Alabama, to Louisiana and to Texas.

The Southern Pacific Railroad Company takes 20, and the Queen and Crescent Line 18 cars daily. The present openings, when all are in full operation, have a capacity for about 2500 tons daily. The coke-ovens, of which 300 are being built, about half a mile from the mines, will be charged with the screenings from the No. 2 seam and coal from the No. 9 seam. The Thomas furnace coke-plant, of 150 ovens, near Ensley City, will be supplied with coal from these mines.

At the present time, March 1st, 1888, 800 men are employed in and about these mines and on construction-work.

The Saint Clair Coal Company, which is at Ragland, is at present

the largest coal operator in Saint Clair County. During 1887, its production reached 52,141 tons of coal, of which 15,047 tons were consumed in the manufacture of coke.

The coal-bed here is 3 feet thick, and the coal is adapted for steam, coke, gas, forge and domestic purposes. The company employs from 40 to 55 men, and in 1887 the mines were worked 275 days, the product, excepting that portion made into coke, being shipped principally to Atlanta and Rome, Georgia; and to Anniston, Alabama.

The Wolf Creek Coal Company's mines are located at Corona, in Walker County. These mines were operated during January, 1887, but were idle the remainder of the year; the total production was only 530 tons. The coal-bed shows a thickness of 3 feet 6 inches, and the coal is adapted for domestic, steam and gas purposes. An analysis shows the following composition:

Ash,	1.903
Fixed carbon,	58.611
Volatile matter,	37.735
Sulphur,	1.953
	100.402
Per cent. of coke,	60.714
Specific gravity,	1.317

The company employs 24 men, and the coal is shipped to Memphis, Mobile and New Orleans.

The mine of the O'Brien Coal Company is located on Wolf Creek, 1 mile west of Corona. The bed becomes a little thinner than it is at Corona and Patton to the east. It, however, still averages nearly 4 feet thick. The mine is opened by a slope 100 feet in length. During 1887, 10,500 tons of coal were mined, the mines being idle 5 months on account of strikes. About 45 men were employed on the average, and the entire product of the mines was shipped to New Orleans, Mobile, Memphis and Montgomery for steam, gas and domestic purposes.

The Virginia and Alabama Mining and Manufacturing Company is at present the largest operator in Walker County. It owns six mines in the vicinity of Corona, all located on the Georgia Pacific Railway; three of the mines are operated by the Company, and the remainder are leased to other parties.

The Patton mine is located three-quarters of a mile back from Patton station, 1 mile east of Corona. The mine is opened by a

drift in the Corona coal-bed, which here varies from 3 feet 8 inches, to 4 feet 4 inches in thickness, and contains a slate parting ½ to 1½ inches thick. This mine employs 130 men, and but for the general strike, which occurred in this district on February 1st, 1887, and lasted 5 months, the output would have doubtless been nearly doubled. The Coal Valley mines, Nos. 6 and 7, are located 2 miles back from Day's Gap station, 5 miles east of Patton. There are also drift-openings on the same seam as at Patton, the thickness being also about the same, averaging over 4 feet. This coal is shipped, run of mine, for steam purposes. The coal-bed through both these mines lies nearly level, and no faults of any kind have been encountered. Mr. David Kirkwood, formerly of the Corona mine, has charge of the underground work, employing 85 men.

The Coal Valley mines, Nos. 4 and 5, are operated by Donaldson & Co. Their mines are leased from the Virginia and Alabama Mining and Manufacturing Company, and are being worked in the Corona seam. About 160 tons are shipped daily, employing 50 men.

The Day's Gap mine, also leased from this Company, is operated by Norwell & Co. They ship run of mine coal, employing 26 men, putting out 60 to 70 tons daily.

The coal from the Corona bed in this Company's mines is particularly adapted for domestic and steam purposes, but is also used for the manufacture of gas. An analysis shows as follows:

Ash,	7.36
Fixed carbon,	50.69
Volatile matter,	41.12
Water,	0.83
	100.00
Sulphur,	0.42
Specific gravity,	1.32

This coal is shipped, principally for domestic and steam purposes, to Atlanta, Birmingham, Vicksburg, Memphis, New Orleans, Shreveport and points in Texas; also to various railway lines, as the Maryland and Ohio railroad, Georgia Pacific railway, Southern Pacific railway and Texas and Pacific railway.

The Deer Creek mines of the Black Diamond Coal Company are located at Patton Junction, in Walker County. These mines were operated during the early part of the year, by Turner, Morris & Co. who produced 3000 tons of coal. The Black Diamond Coal Company after taking charge of the mines produced 9500 tons, making the total production for the year 12,500 tons; the coal-bed worked

at the Deer Creek mine averages about 4 feet 6 inches in thickness and underlies several squares miles of territory. The coal is used for steam and domestic purposes; the following analysis will show its character:

Ash,	6.00
Fixed carbon,	68.34
Volatile matter,	22.15
Sulphur,	1.85
Water,	1.66
Total,	100.00

The mines were only run on half time during the year, fifty men are employed at the mines; the wages earned by the miners average $2.50 per day, and day hands earned $1.25 to $1.50. Shipments were made to New Orleans, Mobile, Birmingham, Atlanta and other points in the South. The average price of the coal delivered on cars at the mines during 1887 was $1.25.

Among the newer companies that have been organized in Walker County, the Corona Coal and Coke Company is probably the most important. This Company is operating one of the largest mines yet opened in Walker County, at Corona station, 55 miles west of Birmingham, on the Georgia Pacific railroad. It owns some 1000 acres of coal property in the neighborhood, and has made a double drift-opening into what has become known as the Corona bed, which averages about 4 feet thick, although in parts of the workings only 3 feet 8 to 10 inches is found. The coal is of fine quality for steam and domestic purposes. It is a hard splint-coal and stands handling well, and can be conveniently stocked in yards during the dull summer months for winter use, thus enabling all the companies operating in this bed to run steadily during the summer months, getting advantage of summer freight-rates. Its qualities as a coking coal have not yet been satisfactorily established. The coal lies very flat in the hill, giving a level haul to the mouth of the mine. No fault of any kind has been encountered in the bed, thus enabling economical work to be done. The present capacity is about 400 tons daily, which can be increased on short notice. An addition has recently been built to the tipple to permit coaling of engines on the road. At present, February 20th, 1888, 125 men are employed.

Another new operation in this county is the Kansas City Coal and Coke Company, whose Carbon Hill mine is located at Carbon Hill, in Section 25, Township 13, S. Range 10 W., Walker County. Active operations were begun in December, 1887, and 2000 tons were shipped during the month. This mine is opened by a shaft 62

feet deep to the bottom of the coal-bed, and is equipped with hoisting machinery with a capacity of 50 tons per hour. The estimated working capacity of the mines is about 400 tons per day. The coal-bed worked is the Jagger bed, 4 feet 8 inches thick, with a seam of slate 6 inches in thickness and 8 inches from the top of the bed. The coal-bed is said to underlie an area 20 miles in length by 3 miles in width.* This Jagger coal is claimed to be the best coal in the State for steam and grate purposes. At present three-fourths of the production is consumed by the locomotives of the Kansas City, Memphis and Birmingham railroad, on the line of which the mine is situated, the remaining one-fourth being shipped to Memphis for domestic uses. The mine employs 75 regular miners, and works regularly every working-day on 10-hour shift.

In the vicinity of Tuscaloosa, in Tuscaloosa County, a number of small operators have been working mines for some years, supplying the local demand at the city of Tuscaloosa. The most important of these mines is that of A. Durie, located 2 miles from the city, which produced during 1887 a total of 3500 tons. Other small mines in this vicinity produced 1750 tons more. The Alabama Insane Hospital, at Tuscaloosa, also operates a small mine for the supply of fuel for the hospital. This mine is located 2 miles north of Tuscaloosa, on the Warrior River, producing during 1887 about 3000 tons of coal, which was all consumed at the hospital. The bed is 2 feet thick and is said to furnish a good gas coal. Tests show a yield of gas of 4 cubic feet per pound of coal. This mine was worked steadily throughout the year, about 4 men being employed.

The most important coal operation in Shelby County is that of the Montevallo Coal and Iron Company, located at Aldrich. During 1887 this company produced 36,355 tons of coal, all of which was shipped to points in Alabama, Georgia, Mississippi and Louisiana for domestic purposes. The coal-bed here averages about 2 feet thick, and the coal is particularly adapted for burning in grates as a house coal. The mines were operated 260 days in 1887, and 75 miners were employed in them.

The wages paid to miners, and the price at which coal is sold at the different mines in the State, vary according to local conditions, probably more so than in any of the States producing coal from the Appalachian coal-field. A number of interesting statistics, showing averages by counties, is exhibited in the following table:

* The Jagger seam is probably the same as the New Castle, and therefore its area is very much greater than above given.

Average Wages, Days Worked and Price by Counties in 1887.

Counties.	Average wages of miners per ton of coal mined.		Average selling price of coal at mines.		Average number of days worked.
	Lump.	Run of Mine	Lump.	Run of Mine	
Jefferson	$0.70	$0.50	$1.20	$.....	244
Bibb	.70	.55	1.50	.....	250
St. Clair		.40	.....	1.15	275
Walker	.72½	.55	1.35	.....	161
Tuscaloosa	.87½	.....	.....	.....	310
Shelby	1.00	.....	2.50	.....	260
Average for State	.71	.51	$1.274		250

Statistics have recently been collected by the Birmingham *Age* which exhibit the present and prospective furnace consumption of coal in the Birmingham district, also the present capacity of a number of coal-mines now being operated, and of others being opened, and the coal-acreage which is controlled by a number of the larger companies. I am indebted to the *Age* for the following facts:

The furnaces given in the following table, showing the consumption of coal, are those already in blast or in course of construction. Several projected furnaces and other industrial establishments, which will undoubtedly be built, are not included in the list:

Local Consumption.

Furnaces.	Tons of Coal consumed Daily.
Mary Pratt, 1,	180
Rolling Mills,	200
Alice and Ensley, 6,	2,600
Sloss, 4,	1,200
Williamson, 1,	200
Eureka, 2,	600
Thomas, 1,	300
Woodward, 4,	600
Bessemer, 2,	1,200
Trussville, 1,	300
Edwards, 1,	150
Gadsden, 2,	500
Sheffield, 5,	1,500
Anniston, 2,	600
Total for furnaces,	10,130
Railroads,	1,500
Commercial purposes,	3,000
Mechanical "	1,500
Total,	16,130

To supply this demand, the following mines are in operation. The capacity of the mines is given, and it must be remembered that the average output (daily) will fall below the maximum capacity of the mine:

Mines.	Capacity, Tons.
Pratt,	3,000
Blocton,	600
Henry-Ellen,	800
Coalburg,	600
New Castle,	200
Mines at Warrior,	400
Montevallo,	200
Other mines in the Warrior field,	1,000
Total present capacity,	6,800

The following mines now being opened will increase the output as follows:

	Tons.
Eureka,	500
Blocton,	1,400
Woodward,	300
De Bardeleben's Blue Creek,	500
Alabama Connellsville,	200
Total prospective,	2,900

The opening of the new mines, as given above, will make a grand total daily output of coal in the Warrior and Cahaba fields of 9700 tons.

The present and prospective local consumption of coal is shown by the tables above to be 16,130 tons. The shortage in supply is, therefore, 6430 tons per day. To supply this shortage there is an urgent demand for the opening of more coal mines. Of the vast body of coal-lands in Alabama, 1,000,000 acres still belong to the government and are open to homestead entry. The bulk of the public mineral lands lie in the Warrior coal-fields. The coal companies now operating in Alabama own the following amount of coal lands:

	Acres.
Pratt Company owns,	75,000
Blocton Company owns,	30,000
Henry-Ellen Company owns,	16,000
Coalburg Company owns,	15.000
Milner Company owns,	5,000
Companies at Warrior own,	16,000
Carried forward,	157,000

	Acres.
Brought forward,	157,000
Eureka Company owns,	10,000
Woodward Company owns,	15,000
De Bardeleben Company owns,	20,000
Samuel Thomas Company owns,	3,000
Montevallo Company owns,	5,000
Alabama Connellsville Company owns,	3,000
Other companies,	20,000
Total,	233,000

The De Bardeleben Coal and Iron Company's lands noted above are located in the vicinity of Bessemer, in Jefferson County, and their operations at the Blue Creek mines and elsewhere bid fair to be almost as extensive as those of the Tennessee Coal, Iron and Railroad Company.

Their first half-dozen openings were unfortunately located and have been abandoned, but others have been successfully driven—one to a distance of 1200 feet, promising some 500 tons of coal a day by the last of March, when it is expected one of the company's two furnaces will be ready to go into blast. During the latter part of the year a slope was being sunk, but had not been completed up to January 1st. The company proposes to consume the entire output of their mines, which it is estimated will range from 2000 to 3000 tons per day, inside of 12 months, at the furnaces located at Bessemer.

The average thickness of the bed is 7 feet, and the company reported that the area underlain by workable coal is 8 miles long by about 2½ miles wide.

The following are analyses made by Mr. A. F. Brainerd of coke made from the company's coals:

	No. 1.	No. 2.
Ash,	10.490	6.200
Fixed carbon,	85.776	92.327
Volatile matter,	3.040	.400
Sulphur,	.544	.823
Water,	.150	.250
	100.000	100.000

No. 1 coke was made from outcrop-coal, while No. 2 was made from coal mined under a good solid roof and is considered a more representative specimen. This Company has under way the construction of 150 coke ovens and will build as many more when these are completed.

All of the above developments are being made within 20 miles of

Birmingham. About twice as far away, in the untouched field of Murphree's Valley, which is very rich in both coal and iron, there are being located a number of mines with a view to very extensive operations.

The Bessemer Coal and Coke Company, made up largely of the same people as the De Bardeleben Company, are opening 7 drifts on Coal Creek, near Parksville, on the Kansas City, Memphis and Birmingham railroad, in Jefferson County, some 25 miles from Birmingham. These openings are being made in the Pratt seam, and it is expected that coal shipments will be begun in June, 1888, at the rate of 800 to 1000 tons daily. The mining engineers of the two companies say that in two years, according to their present plans, they will be able to take out 5000 tons daily. It is expected to send all this coal to Bessemer and Sheffield for local and furnace-consumption.

An analysis made by Mr. A. F. Brainerd of 12 tons of the company's coke shows the following composition:

Ash,	10.545
Fixed carbon,	87.299
Volatile matter,	.803
Sulphur,	1.195
Water,	.157
	100.000

The Alabama Connellsville Coal and Coke Company, a new concern, neighbor to the De Bardeleben Company, will be able to mine 500 tons a year from now if its present projects turn out fairly. Its coal-bed is reported to be, in places, 11 feet thick and has but 1 parting. The dip, however, is such as to make mining more difficult than elsewhere in the district. It is reported that a considerable coal property will be developed, during the year, in the West Warrior field, mainly in Walker County, but how this will affect the iron industry is as yet largely a matter of conjecture, no satisfactory smelting coke having been made from any of the coals of this region.

The Eureka Company has bought the Helena property of the Tennessee Coal, Iron and Railroad Company, and will erect 100 ovens. The Cahaba Coal and Mining Company, which, like the Eureka Company, is operating in the Cahaba field, has made a beginning on 400 ovens, mainly to supply the Woodstock furnaces, in and near Anniston, and the Pioneer furnace near Birmingham.

The Woodstock Coal and Iron Company recently bought an in-

terest in the Cahaba concern. The Pioneer Manufacturing Company's contract runs for only one year from the completion of their furnace, after which period it is expected that it will mine its own fuel somewhere on its extensive coal-property.

The Sheffield people seem to believe in the qualities of the West Warrior coal for iron-making. The Alabama and Tennessee Coal and Iron Company has broken ground for 300 coke ovens at Jasper, Walker County. Several coal operators in the same county have contributed to build 6 experimental ovens at the same place, after the patent of Mr. Koener, who is superintending the construction, and on the general plan of the Coppee oven.

The Tuscaloosa Coal, Iron and Land Company is making extensive explorations in the vicinity of Tuscaloosa.

During the year much has been said relative to the Huntington Inter-oceanic railway enterprise, which proposes a southern outlet for the coal and other products of the Mississippi Valley. Birmingham coal is delivered in Mobile and along the coast of the Gulf of Mexico, at $1.37½ per ton, and may be delivered in barges at any port on the Bay of Honduras at $2 50 to $3 a ton. English coal is sold annually, to the amount of about $50,000,000, at from $8 to $12 a ton, in ports on the Gulf of Mexico, and to the islands and neighboring coasts of the three Americas and Australia; and English coal and iron are supplied to steamers and railways along the whole Pacific coast, from San Francisco to Cape Horn. The cost to consumers is enormous, but it would be cut down to about one-third the present rate if enterprises of United States citizens in Central and South America were fostered and encouraged by the Government of the United States.

STEEL RAILS, AND SPECIFICATIONS FOR THEIR MANUFACTURE.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Buffalo Meeting, October, 1888.)

HAVING had some twenty years' experience in trying to make good Bessemer steel rails, and now devoting my thoughts and energies to seeing that other people seek the same end, I venture to lay before the Institute some of the conclusions deduced from my practice and observation.

Those who know only the Bessemer manufacture of to-day, cannot appreciate the many changes which have taken place in steel-rail making. The first steel rails used in the United States were imported by the Pennsylvania Railroad Company from England. The first rolled in this country were made at the North Chicago rolling-mill from steel made at Wyandotte, Mich., on May 24th and 25th, 1865. But this was simply an experimental rolling of six rails. In 1867, some tons were rolled in the Spuyten Duyvil, N. Y., rolling-mill, from steel made at Troy, N. Y. But it was not until August, 1867, when the Cambria Iron Company, of Johnstown, Pa., arranged with the Pennsylvania Steel Company, of Harrisburg, Pa., to roll the steel of the latter into rails—the rail-mill at Harrisburg not being completed—that the making of Bessemer rails in America was put upon a commercial basis.

It was my fortune to have charge of the steel department of the Cambria Iron Company at that time; and, having been ever since engaged in rail-making, I know the accepted practices of the business from that period, the difficulties encountered and the measures adopted to avoid or overcome them. The word Steel *then* conveyed a very definite conception to the minds of all men. The character and peculiarities of its behavior under treatment, either hot or cold, were expected to be like those of tool-steel, as that had been observed while under the manipulations of the maker, or afterwards of the smith. And the one thing most earnestly impressed upon every one's mind, and emphasized as the very essence of good workmanship, was, that steel would bear only the most moderate heat. In the converting-house, all the possible practices of crucible-steel teeming were introduced. The ingot-moulds were carefully brushed out, heated and smoked before being used. When the steel was teemed, all doors and windows of the casting-house were closed, etc., and time was not spared on any of the details. This was the English practice; and we in America followed it.

At first, steel rails were rolled in mills which had been designed for iron rails. Other rolls were used, and the number of passes was increased, making the reductions very gradual. But even with these easy reductions, at the low temperature at which the metal was worked, and with the weak machinery, breakdowns were distressingly frequent. In May, 1868, A. L. Holley, who then had charge of the Pennsylvania Steel Works, completed and started that company's steel-rail mill—the first mill in America especially designed for that work. In designing this, Mr. Holley accepted the 21-inch

Cambria mill as affording the best type of the Fritz three-high mill, and, while increasing the diameter of the rolls and otherwise strengthening its parts, largely followed it, even to the general plans.

But steel rails were not immediately accepted as the rails of the future. For a long time the two leading English engineering papers, *The Engineer* and *Engineering*, carried on a controversy over the merits of iron *versus* steel rails. And the railroad-people suggested, early in the day, that if steel rails were to be adopted, they must be obtainable at a low price. But we could not keep on breaking coupling-boxes, rolls, housings and engines, and give our friends cheaper rails, and at the same time make money for our companies. No one dreamed that the business would, in the smallest degree, approach what it has since reached. Had an engineer proposed making the necessary outlay (even had he known how) to build a mill which would have safely handled the metal as then treated, he would not only have been denied the money, but the services of some safer man would have been sought by his prudent and wise directors. To name $600,000 as the cost of a rail-mill would have endangered an engineer's personal liberty. So, one day, some bold fellow queried, "Can't we get the stuff a little hotter?" It was tried, and a little, after a while, became a good deal; and that, hotter still, until many tons of blooms have been drawn from the heating-furnaces with the cinder running off them.

In the early days, all blooms were allowed to cool and were carefully cold-chipped before being charged into the re-heating furnaces. After the drawing of one heat, and before the charging of another, the doors of the furnace were always raised and the furnace cooled down. Then the heat was brought up very gradually and plenty of time was taken to allow the steel to "soak." It was expected to produce but 50 per cent. as many steel as iron rails, and all employees working by the ton were paid twice as much for steel as for iron. But the railroads wanted cheaper rails, and more Bessemer rail-mills had been built and were building. They were all looking for work and were desirous of meeting the views of their friends, the consumers. The Bessemer managers were not asked if they *could*, but were told: "You *must* manage to lessen your costs and, by all means, get out more product." Truth compels me to say that they did not need much urging on this last point. And as the certainty became manifest that steel rails had come to stay, they were permitted to alter, add to and strengthen their plants.

Well, the rush was kept up, slacking when dull seasons came, to be renewed with redoubled vigor when the tide again set in. Change followed change, in rapid succession, in the details of the manufacture at all its stages. No doubt many of the earlier practices of both the converting-house and the rolling-mill were carried too far; and some of them were utterly without use.

In 1876, I had the honor of presenting to the Institute "A History of the Bessemer Manufacture in America" (*Trans.*, v., 201). In it, I endeavored to tell tersely of the rapid changes which had occurred in the art up to that time. It is not necessary to repeat them; suffice it to say again that in all its branches they came very rapidly and have not yet ceased.

Remember, during all this time the makers have been trying to make good rails, harassed beyond measure with complaints and seeking most earnestly to obviate them. But let the complaint be never so loud, the seller offering rails at 25 cents per ton less would, as a rule, take the order.

Such has been the history of the business.

When steel rails were first put down, the railroad-traffic had increased to so great an extent and the quality of iron rails had so fallen off, that it was impossible for the roads to keep up their tracks. The results from the first steel rails were so wonderful that, as a rule, no very careful records of their wear were kept. After a time this was changed; but by that time the "lame ducks" had been eliminated, and we had "the survival of the fittest." The poorest steel rails had yielded results so much beyond those obtained from iron rails, that their failure had not caused much comment. But when the record-period was reached, every failure caused a shock of horror; and the question was asked: "Why can't you make rails like the early ones?"

While claiming that injustice has been done to later rails, I acknowledge at the same time that many of them have not been as good as the best of the earlier ones.

I have candidly admitted some of the changes on the part of the makers. How has it been on the part of the consumers?

The weight of engines and cars, speed of trains and amount of traffic have been increased. How much?

The railroad-engineers and managers have been spurred on by the same character of questions as their brethren of the Bessemer faith. "Can't you give us cheaper service and greater tonnage?"

They also have answered, every time, "Yes."

If rails costing $30.00 per ton do not last as long as those which cost $125.00, and bear at the same time many times as great an annual tonnage, how great is the proportionate loss to the purchaser?

But no one desires to turn backward. The rail-makers do not want to confine their mill to a boasted "annual capacity of 30,000 tons." Neither do the railroad managers long for light engines, small cars and scanty traffic. The country has grown. The world moves. Retrogression is impossible. The conditions being as they are, the duty of all parties is to work in harmony, and, drawing conclusions from the experiences of the past, seek to produce rails which will yield fair service under the present requirements, without greatly increasing their cost.

The first duty falls upon the railroad-engineer in designing his section. If that is bad, the steel-maker will be heavily handicapped in trying to furnish a satisfactory rail. I suppose it is hopeless to expect the adoption of standard sections. We are seemingly no nearer it than when A. L. Holley read his admirable paper on that subject at the Philadelphia meeting of the Institute in 1881 (*Trans.*, ix., 360). I will not, therefore, at this time, venture to submit any designs. I will leave that duty to some one with the bright hopes of youth imprinted upon his brow. But I will say that I am fully convinced that great mistakes have been made in the designs of the heavier sections, and believe that the unsatisfactory results obtained from many of them have been primarily due to the proportions of the rails. One of the most prominent chief engineers of the country said to me not long ago, that he supposed his road would lay 80-pound rails next year, but he was at a loss what section to adopt; for if the 80s gave as much poorer service as compared with their 65s, as the latter had done in comparison with their old 60s, he guessed they would have to adopt a 90-pound rail for the following year. This states it broadly; but it is a generally admitted fact that the increased sections have, as a rule, been disappointing in their wear. I am convinced that this has been largely the fault of the sections themselves.

Increased weight of rolling-stock, greater speed, augmented traffic, etc., demand heavier and stiffer rails. The exhaustive experiments of O. Chanute, presented to the Institute in 1881 (*Trans.* ix., 578), demonstrate the rapid augmentation of the crushing-strains as the wheel-weight is increased. He found that with "11,000 to 12,000 pounds weight upon a locomotive driving-wheel of about five feet

diameter, the pressures were generally 35,000 to 40,000 pounds to the square-inch, although they occasionally ran much beyond this; but with 14,000 pounds on a driver, the pressures became from 50,000 to 80,000 pounds to the inch, or beyond the elastic limit, even of steel. Under empty freight-cars, with say 2400 pounds on a 33-inch wheel, the pressures were generally 20,000 to 30,000 pounds per square inch; but with a car loaded with 11 tons, increasing the weight to say 5150 pounds per wheel, these pressures became about 35,000 pounds to the inch; while if the car was loaded with 20 tons, thus giving 7400 pounds to the wheel, the pressures increased to 50,000 or 60,000 pounds to the square inch." And he added: "As we increase the weight upon our cars, therefore—and I believe this to be the correct and inevitable practice—we must be prepared to find our steel rails wear out faster than they hitherto have done."

But is increasing the depth of the metal in the heads of the rails the proper way to meet this condition of things? One reason given for increasing the depth of metal in the rail-head, has been the provision of more metal for wear. But is this economy realized? When a given amount has been worn off, the rail is so rough that it must come out of the track. English double-headed rails were expected to be turned after the first head was worn out. But was it ever done?

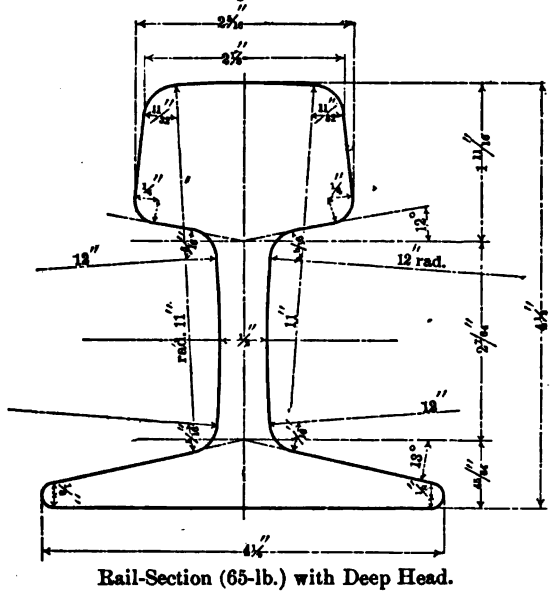
The larger the mass of metal when finished, the hotter it will be, provided the same length of time has been spent in putting it into shape. And every steel-maker knows the effect of cold-finishing, that is, having the steel at a low temperature when the final reductions are made. The smaller the section rolled on a given mill, the deeper will the effect of the compression of the rolls penetrate, and the finer will be the grain of the steel.

I would favor increasing the width of the head more than its depth. I know it has been argued that, owing to the shape of the wheel-tread, only a given amount of bearing can occur. But is this true? And how long does the wheel-tread maintain its original form?

In discussing the proper shapes for rail-heads, importance has been attached to the forms of the tires which are to pass over them. As just suggested, at which stage of the tire's life will this have the most effect? One of the principal tire-makers of the United States tells me that by the views of their customers they are compelled to

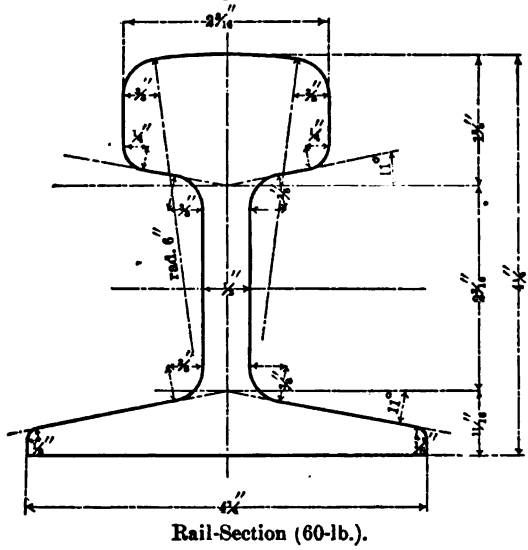
keep rolls for 115 different sections. My friends, the rail-makers, must not think they have a monopoly of all the trouble!

Fig. 1



Rail-Section (65-lb.) with Deep Head.

Fig. 2

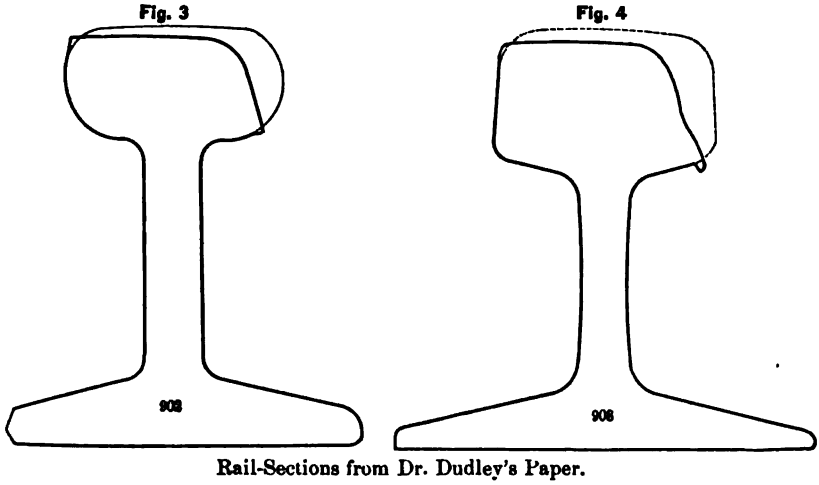


Rail-Section (60-lb.).

Since brittle rails must be avoided, as well as rapidly-wearing ones, the engineer, when designing his rail, must not lose sight of

the importance of distributing his metal so as to avoid, as far as possible, cooling-strains.

As I have already said, I do not now propose presenting *my* ideal rail-section. But to support my objection to deep heads, rolled on existing mills in the ordinary manner, I give two sections. The first (Fig. 1), a 65-pound rail, is represented by many thousands of tons now in service, and has been represented by many other thousands of tons, whose days of usefulness are over. Not that these rails can be said to have failed; but they have not given the service expected when the design was adopted. The second section (Fig. 2) is that of a 60-pound rail which has been in the tracks of one of the



Rail-Sections from Dr. Dudley's Paper.

roads using also the first section, for many years, and insists upon remaining there, to witness the coming and departure of the "big fellows." I may be wrong, but these appearances seem to be on my side. My conclusions are not formed, however, from this single case. They are based upon the observations of many years.

In discussing Dr. Dudley's paper (*Trans.* ix., 321), in 1881, I called attention to the fact that rails of the closest similarity in chemical composition differed widely in wear. In fact, one of Dr. Dudley's best (No. 902) and one of his poorest rails (No. 903) were chemically practically the same; but the best was a rail of round and shallow head, the other a deeper-headed pattern. As these so fully sustain my position, I give them with their history.

"No. 902 (Fig. 3), Steel of 1871, was on high side of 5-degree curve, north rail, north track, in Gable's Cut, near Summer Hill. In

service from August, 1871, to July, 1879—7 years, 11 months. Grade, 39.6 feet to mile. Tonnage, 40,061,230 tons."

"No. 903 (Fig. 4), Steel of 1876, was on high side of 5-degree curve, south rail, south track, west of Bridge No. 1, Summer Hill. In service from August, 1876, to July, 1879—2 years, 11 months. Grade, 39.6 feet to mile. Tonnage, 21,504,824 tons."

Their analyses were as follows:

	No. 902.	No. 903.
Carbon,	0.322	0.355
Silicon,	0.026	0.029
Phosphorus,	0.077	0.108
Manganese,	0.492	0.490

If the manner of supporting the joints is not right, the rail-makers must not bear all the blame when they go down. I think that most railway-engineers will unite in saying that as yet a perfect joint has not been attained. Some are much better than others; but as to which is the best, they are far from being in accord. With so many bright minds investigating this subject, we may expect the best possible results to be reached in time.

We will assume the section selected to be the best possible; it now remains for the maker to furnish a good rail rolled to it.

The character of the permanent way of the railroads of the United States is improving each year. Consequently, the demand that the mills shall deliver their rails well finished and straight in all directions is much more imperative than in the past. Of course, absolute accuracy in so gross a product is both unnecessary and impossible; but practical accuracy is attainable.

I consider it of the utmost importance that most of the straightening shall be done while the rails are hot. In other words, the hot-straightening should be conducted so as to leave the minimum work for the cold press. Every blow of the gag is a bid for a break. The harder the steel the greater the danger. Moreover, gagging is apt to take out one bend by putting in two others, endeavoring to have two wrongs make a right.

To bring the breaking danger down as low as possible, the cold-straightening should be done before the rails are absolutely cold. At the same time, if the rail goes under the cold-press too hot, the steel will not possess elasticity, and each blow will leave a dent; and if excessively applied, the rail will be either "wavy" or "lumpy." Such rails will of course make a rough track.

Some mills, in taking side-bends out of their rails, apply the gag

to the flange. I cannot approve of this, and believe that it greatly increases the danger of broken rails.

After the rails are delivered to the track-man, they should not be carelessly thrown from the cars. This was not the practice with the early steel rails.

The drilling should be accurate; and if proper drill-presses and drills are used, it can be made so. If the holes are to be anything from $\frac{1}{32}$ under to $\frac{1}{16}$ over size, I do not see any use in naming the diameter in the specifications.

So long as the purchasers of rails exact a guaranty from rail-makers, I think the chemical composition of the steel should be largely left with the latter. But if the purchaser believes that carbon is the best hardening-element for steel, it is not unreasonable to ask for as much as the maker is willing to put in, knowing what proportions of other elements his metal will contain, and still guarantee his rails. No doubt, as the sections are increased, harder steel can be safely used.

The mere replacing of a broken rail with a whole one, may fall far short of the damage sustained by the road on the track of which the accident has occurred—even leaving out of consideration the danger to human life. Therefore the purchaser has a right to insist upon some precautions against such disasters. And these precautions, if correctly taken, are also in the interest of the maker. Steel rails are made very rapidly, and the demands of the trade necessitate that they shall be made very cheaply. The workmen are paid by the piece; and, while generally making good wages, they must produce a large tonnage to realize them. No matter how desirous the general management may be of producing good work only, it is very necessary that safeguards should be provided. Most makers have these in some form; but there are mills where the chances are taken.

The tests which I prefer are those I used for fifteen years at Troy. My experience gives me confidence in them. I do not wish to say that Troy rails have never broken in service; but such accidents have been infrequent, and could almost always be traced to individual mechanical causes.

This plan of making tests has the merit of furnishing a check on the grade of the steel early in its manipulation; and I consider it more convenient than any drop-test, and at least equally efficient.

As may be generally known, the Troy works have for years made a very wide range of Bessemer steels, from 0.05 to 1 per cent. of

carbon; and we were fortunate enough to have considerable reputation for our success in so doing. On the higher grades, where great accuracy was required, it was my practice to have a test-ingot taken to represent every 15-inch ingot cast. This accomplished two results: it made the workmen careful; and it let us know of any variations which might occur. I do not think this extra precaution necessary in making rail-steel, at any works where ordinarily good practice prevails,—and I should most certainly discourage a client of mine from contracting for rails with others.

As has been proved by the fracture of many ingots, the metal in cooling, if the ingot is left upright until the interior steel sets, will form a funnel-shaped cavity in its top-end. But if the ingot is thrown upon its side before that metal has solidified, this cavity will extend lengthwise; the distance being limited by the condition of the interior steel. Hence it is manifest that ingots should not leave an upright position before the metal has set sufficiently to prevent this cavity from so extending. Everything is against this lengthwise-defect being taken out by the subsequent rolling of the ingot, and it will most probably cause "pipes" and "cold-shuts" in the rails; but if the cavity is maintained at the top end of the ingot, it can be cut off. Again, if ingots are drawn too soon from the pit and thrown upon their sides, there is danger of the crust, which has formed at the top-end, breaking and permitting the interior liquid steel to escape, or "bleed." That will certainly make a pipe.

It is from sound and compact ingots only that we hope to produce good rails. Such ingots can be made only by care in casting them. Therefore the careful steel-maker will not only use good moulds, but also exercise a close supervision over the manner of pouring the heats. When from any cause this is not done, or cannot be done, the resulting ingot should not go into No. 1 rails.

Nearly every manufacturer uses somewhat different-sized ingots; and frequently they vary in their shape. Of course, each one follows the practice which under the controlling circumstances seems to yield the best results. It is well-known that the same length of ingot will not always roll equally well. I have found, when the metal was cracking badly in the blooming-rolls, that good results could be at once obtained by pouring the ingot shorter. Of course this is easily explained, and points to the necessity of closely watching and controlling the temperature of the "blow" in the converter. I fully believe that a strong influence on the quality of the resulting rail rests at this very point.

Since every ingot, if properly handled, has more or less of a cavity at its top-end, the bloom rolled from it will be piped or spongy at that end. To be certain of having a sound rail made from the upper part of the ingot, a sufficient length must be cut off to remove this spongy steel. These pieces need not necessarily be treated as scrap; there being many purposes for which they will answer.

It is not necessary for me to say here that care should always be exercised in heating steel. As the carbon is increased, so is the danger. I believe more unsatisfactory rails can be traced to overheat in the furnace than to any other one cause. I commend the paper of our fellow-member, William Metcalf, on "Steel; its Properties; its use in Structures and in Heavy Guns," read before the American Society of Civil Engineers, March 2d, 1887, to the careful consideration of every steel-maker and user. It is in my judgment worthy to be considered a text-book on the treatment of steel.

My investigations of the service of thousands of tons of rails, and the analyses of many hundreds of them, have shown the greatest variation in the wear of rails of the same section and chemical composition. This being so, there must be some physical cause. Can we find a chemical reason for rails showing "soft" in wear, having the following chemical composition?

Carbon,	0.39
Sulphur,	0.059
Phosphorus,	0.085
Manganese,	0.722

If so, why did another make, in the same track and under seemingly the same conditions, analyzing as follows, wear "hard?"

Carbon,	0.40
Sulphur,	0.064
Phosphorus,	0.080
Manganese,	0.779

I could multiply these instances to an indefinite extent, but will not take up time.

Our Bessemer friends are all right on their chemistry. They know a great deal more than the people who made those early good rails, and it is not in that direction that investigation is most needed.

As I said early in this paper, every rail-maker wants to give his customers good rails. Now I honestly believe it is to his interest that the purchaser should be represented by intelligent inspection. No matter how good the mill-organization may be, the men all work

by the ton, and do not always realize the importance to their own interests (which are the same as those of their employers) that only good work should go out. Hence the right kind of inspection is of assistance to any mill.

If I were a purchaser of rails, I should draw fuller specifications than these, which I now have the honor to present to you, and I should accept all the risk of the results. But railway-managers are not yet willing to assume this position. I have therefore endeavored, while not relieving the makers of any responsibility, or introducing novel practices, to assist the railroads in obtaining better and more uniform rails.

It is recognized as the commercial rule that rail-makers should give a guaranty with their rails. In these specifications, I have embraced such an one as is given by some of the largest makers in the country, and under which they have sold rails for many years. It seems to me to be fair in its provisions; and I believe the other requirements of my specifications will tend largely in the direction of making the sellers safe in the guaranty. If this is true, the railroads will of course receive satisfactory rails, and everybody ought to be happy.

Specifications for Bessemer Steel Rails.

Section.

Section 1. The section of the rail rolled shall conform to the template furnished by the railroad company with an allowance in height of $\frac{1}{64}$ of an inch under, and $\frac{1}{32}$ over, being permitted in a delivery of 10,000 tons of rails. The fit of the fishing or "male" template shall be maintained perfect.

Sec. 2. The weight of the rail shall be kept as near to — pounds per yard as is practicable after complying with Sec. 1.

Lengths.

Sec. 3. The standard length of rail shall be 30 feet at a temperature of 60 degrees Fahrenheit. Shorter rails of —— lengths will be accepted to the extent of 10 per cent. of the entire order. A variation in length of $\frac{1}{4}$ of an inch longer or shorter than the above specified lengths will be allowed.

Finish.

Sec. 4. The rails must be free from all mechanical defects and flaws, and shall be sawed square at the ends, and the burrs made by

the saws carefully chipped and filed off, particularly under the head and on top of the flange. In sawing, care must be taken to avoid a flow of steel which will produce a swell on the top of lower flange, as the rail lies under the saw, thereby affecting the fit of the fish-plate.

SEC. 5. The rails shall be smooth on the heads; straight in all directions, both surface and line; and without any twist, waves or kinks; particular attention being given to having the ends without kinks or drop. The hot-straightening shall be carefully done, so that gagging under the cold-press will be reduced to the minimum, and so applied that the rails shall not be made "lumpy."

Drilling.

SEC. 6. Circular holes —— inch in diameter shall be drilled through the web at —— inches from the bottom of the flange. The center of the first hole —— inches from the end of the rail; and —— inches from the center of the first to the center of the second hole; and so on, if more than two holes are required. These holes must be accurate in drilling in every respect, and left without burrs.

Branding.

SEC. 7. The number of the charge, the name of the maker, the month and year of manufacture, shall be marked in plain letters and figures on the side of the web of the rail, in such a position as not to be covered by the fish-plates when laid in the track. If the purchaser prefers, the number of the charge shall be stamped on the end of the rail.

Percentage of Carbon.

SEC. 8. The steel to contain as high a percentage of carbon as the maker is willing to put in and still meet the requirements of Sections 9 and 21.

Tests.

SEC. 9. While the heat is being cast, two test-ingots shall be made; the first from steel going into the first regular ingot; the other from metal representing the last one. These test-ingots shall be 3″ × 3″, and not less than 4″ long. From these, bars at least ½″ square shall be drawn at one heat by hammering. Each bar when cold shall be bent, without breaking, by the blows of a sledge to not less than a right angle. Should one bar from a heat fail and the other stand the test, a third bar may be taken from a bloom rolled

from the same ingot represented by the failed bar. If this stands the test, it shall be accepted in lieu of the failed one. If the makers choose, more than the two test-ingots may be taken, but they must be from the steel of the first and last regular ingots. If this is done and a test-bar fail, another one may be drawn from the duplicate ingot and tested, and if it stands, accepted.

Treatment of Ingots, etc.

SEC. 10. After the ingots are cast, they shall be either constantly kept in an upright position until ready to be rolled, or else so maintained until the interior steel has had time to solidify.

SEC. 11. No "bled" ingots, or ingots from "chilled" heats shall be used in the manufacture of rails under this contract.

SEC. 12. No ingots from badly-teemed heats shall be used; excepting as they shall be subject to the provisions of Section 16.

Cutting of Blooms.

SEC. 13. After cutting off, or allowing for the "sand" or top-end of each ingot, at least 12 inches more of seemingly solid steel shall be cut off that end of the bloom, or partially formed rail; if the latter, then the pieces so cut off shall equal 12″ in length of a 7″ × 7″ bloom; a greater length than 12″ being preferred; and if, after cutting such length, the steel does not look solid, the cutting shall continue until it does.

Heating.

SEC. 14. Care shall be taken to avoid overheating the steel, in shape of either ingots or blooms; and under no circumstances shall a "cinder"- heat be allowed—that is, a heat high enough to cause the cinder to run off the steel as it is being drawn from the furnace. This does not apply to cinder which may be sticking to the underside of the steel, when drawn from a horizontal furnace, or to the bottom of an ingot when drawn from a soaking-pit.

Inspection.

SEC. 15. Inspectors representing the purchaser shall have free entry to the works of the makers at all times while this contract is being filled, and shall have all reasonable facilities afforded to satisfy them that the rails are being made in accordance with these specifications. The makers shall furnish them with the carbon determinations of each heat, if so required.

SEC. 16. The inspectors shall have power to reject rails made from insufficiently-sheared blooms; or from heats, the test-pieces of which have failed; or from badly-poured heats; or from "chilled" heats; or from "bled" ingots. The rails made from insufficiently-cut blooms, if otherwise perfect, to be afterwards received as No. 1 short rails, if sufficient lengths have been sawed off to make an amount of steel equal to the original demand of 12 inches. The rails made from heats, the test-pieces of which have failed, may be accepted as No. 2 rails. The rails from a badly-poured heat may be received as No. 2 rails; but if made from a "chilled" heat or "bled" ingot, to be absolutely rejected. By a badly-poured heat is meant one which, from any cause, has been teemed without the control of the operator. A "chilled" heat is one which, by reason of the steel chilling, has to be either pricked, or poured over the top of the ladle. A "bled" ingot is one from the center of which the liquid steel has been permitted to escape.

SEC. 17. Imperfectly drilled, straightened, or chipped and filed rails shall be rejected, but will be accepted after being properly finished.

SEC. 18. Rails failing to comply with Section 1 will be rejected as No. 1 rails.

No. 2 Rails.

SEC. 19. The requirements for No. 2 rails shall be the same as for No. 1, except that they will be accepted with a flaw in the head not exceeding $\frac{1}{4}$ of an inch, and flaws in the flanges not exceeding $\frac{1}{4}$ inch in depth, and may have been made from an imperfectly poured ingot, or heats from which the test-bars have failed.

SEC. 20. No. 2 rails to the extent of — per cent. of the whole order will be received.

Guaranty.

SEC. 21. The rail-makers to guarantee the No. 1 rails against breakage and unusual wear, at the ends or elsewhere, for five years from the time of delivery to the railroad company. And should any such rails so fail, will upon the return of such failed rails to their works deliver free of cost on cars at their works perfect rails to replace such failed rails; the failure of which is not attributable to improper laying, or want of care after being laid, or unusual circumstances of derailment from failure of other railway machinery or appliances, or negligence of the railroad company's employees. In event of failure at the ends or elsewhere of the No. 1 rails, not ex-

ceeding 10 per cent. of the amount of the contract, before the expiration of the five-years' guaranty (and when the rails in all other respects warrant such a course), the railroad company will cause to be cut off so much of such rails as may be necessary to make perfect rails of them, but in no case leaving them less than — feet in length; the makers to pay in cash for cutting, redrilling, and re-straightening such rails, and the loss in weight so sustained by the railroad company to be made up to them by the makers, on the return to them of the pieces so cut off, in good and perfect full-length rails, of such section as may be agreed upon. The points of delivery of failed rails, ends of rails cut off, and rails to replace the same, or mode of such settlement, may be varied to conform to the peculiarities of each contract.

Discussion.

C. P. Sandberg, London (Communication to the Secretary): By a singular coincidence, I handed in to the Secretary of the Institution of Civil Engineers, London, on the 29th of October last, the month in which Mr. Hunt's paper was published, a paper on the Use of Heavier Rails for Greater Safety and Economy in Railway Traffic, of which a private copy had been sent to a client in America on the 27th. The papers actually crossed each other on the Atlantic, and a *resumé* of Mr. Hunt's appeared in the "Railroad Gazette" and the "Engineering News," which arrived here a week after my paper's departure. Unfortunately there has been some delay in the printing of my paper by permission of the Council of the Institution of Civil Engineers here; and as their regulations do not permit its contents to be given to anybody before it has appeared in print from the Institution, I have been kept in silence hitherto. But I expect the paper will be out soon, and shall then send it freely to American friends. In many respects it touches upon the same subject as Mr. Hunt's; but without prematurely repeating its observations, I will now deal with the question it discusses, in special application to American conditions.

Mr. Hunt's history of rail-making in America is very interesting, and, so far as I have been able to follow it, correct; and many good lessons may be drawn from it, not only for America but also for other rail-making countries. No other country in the world could show anything compared to it; and I am quite convinced that many a European rail-maker would have a good deal to learn from a visit

to America. It is only natural that there should be more perfection in America; for the trade has flourished commercially, while technically, it has had the advantage of the experience of all the European countries; for nothing has been kept back. How could it be, when such men came over as A. S. Hewitt, A. L. Holley, and several eminent rail-makers, all offering to open their doors and show the Europeans their new plants. I regret not having had the opportunity to take advantage of such a liberal exchange of experience by visiting America; but I have had the good fortune of receiving American friends on this side, and of assisting them in getting the information they wanted. I trust that this liberal exchange will continue, and that no political motives or narrow-minded views of any kind will ever prejudicially affect the exchange of opinions between engineers on both sides of the water. Our science is too noble in character to be checked by misplaced patriotism.

Mr. Hunt candidly admits that the rails now-a-days do not wear as well as those made in former days, and gives in explanation facts on the part of both producer and consumer. The principal reasons he adduces are the form of the sections and the want of inspection. I would add a third—the fast driving of both the metallurgical and the mechanical processes.

1. First, as to the section. My aim at establishing standard sections in the early period was not without success, at least as long as rails were imported into America from England. My first publication on this subject, dated 28th January 1870, showed designs for rails up to 80 lbs. per yard. The medium sizes, from 50 lbs. to 60 lbs., were made in very large quantities in both iron and steel, and some still remain in the track. Curiously enough, I have just received a series of sections proposed by Mr. Forsyth, of the Union Steel Company, Chicago, which are almost similar to mine in the head and the flanges. This is after eighteen years' experience. Unfortunately, engineers are the deciding parties; and they have drawn their own sections with a view to economy, placing as much substance in the head as possible, and making the flanges almost as thin as they could draw them—thinking it very clever to give the maker "a job" to roll them. Naturally, no hostile demonstration on the part of the makers was advisable; they had to adapt themselves accordingly, and to construct mills capable of doing what they were directed to do—and they succeeded well enough in the execution. However, the engineer made a whip for his own back thereby, getting in consequence too soft a rail, which would not

wear compared with the older sections, having flanges of at least ⅞ inch thickness. Now they are complaining of the "lost art" of rail-making, and that the big heads do not wear as well as those showing better proportion between head and flange. Two years ago, I gave a paper to the Institution of Civil Engineers here on Rail Joints and Steel Rails, in which I treated the question of over-schemed rail-sections. I am therefore only too glad to see that the subject is brought up again in Mr. Hunt's paper. My forthcoming paper will also deal with the question. I need, therefore, only now endorse all that has been said about the evil produced by wide, thin flanges, which have gone to the extreme in America, and I trust will meet with a speedy correction on the part of engineers by the designing of sections that can be made harder—as hard as the old sections in the beginning of steel rail-making—to enable makers to regain the "lost art" which engineers now lament.

2. With reference to the second cause of inferior rails, namely, want of inspection, I am placed in a somewhat delicate position, and can only say that nowhere in Europe are rails delivered without inspection during the make. Everything depends upon the kind of inspection. Sometimes specifications are overdone and impracticable. In general, the guaranty is not found to be of much value, practically speaking. Buyers prefer to be convinced of having got their money's worth before they pay for the goods.

As for the specification proposed, I can only refer to the one given in my paper before the Mining Institute, and read at the Lake Superior Meeting in August, 1880 (*Trans.* ix, 193), to which I still adhere, and which has served for over a million tons' inspection in my practice. On comparing it with Mr. Hunt's the reader will see the difference, particularly in the applied tests; for while I use the bending-test with sample-ingots, as a preliminary, chiefly to enable the maker to avoid rolling brittle steel into rails, which might break under the falling-test, Mr. Hunt is satisfied with such a preliminary test alone, and does not enforce any falling-test at all. What is more, in his specification, the bending-tests are repeated two or three times, with a chance of ultimately getting one to stand, and having the rails consequently accepted. Mr. Hunt is not the only one abolishing the falling-test or drop-test. I have, in the paper above referred to, reduced the so-called "barbarous" tests so as to admit of greater hardness; but I see no cause to abolish them altogether.

I observe, at the close, the cutting off of blooms 12 inches at the

top-end. Fully appreciating the importance of sound ends, I think such a heavy cut must entail great loss upon the maker; and, worse still, that it will not absolutely secure the object in view, inasmuch as hollow sinkings will not always be detected thereby. I would rather aim to obtain in casting a more solid top to the ingot, through the quality of the steel. To promote this, I would tolerate more silicon, which makes the steel in casting very quiet, and permits hardly any gases to escape. From thousands of analyses, I have found that with 0.10 per cent. of silicon a more solid top is obtained, while no material effect is shown in the testing for physical qualities. I could cite several works on the Continent, using regularly as much as a quarter per cent. of silicon; their ingots are as if they were planed on the top, and need only the cutting off of a 12-inch crop-end of the rail; neither ingots nor blooms being cut at all. In this theory I know that I stand alone, both in Europe and in America, and am therefore quite prepared to meet with objections. My difficulty in practice is to obtain the silicon in uniform proportion. While I sometimes get accidentally a quarter per cent., at other times I get next to nothing, and then I get the hollow top. Regularity in the pig-iron, as to its contents of silicon as well as its heat, would rectify these irregularities in the finished article. This would be well worth the makers' attention, as a means, to ensure both economy to themselves, and sound rail-ends, and satisfaction to the consumer.

To show how much I appreciate the hot-straightening, I beg to refer to a letter on that subject which I sent to *Engineering* ten years ago (10th October, 1879). I congratulate the American rail-makers on their advance beyond the Europeans in this respect. However, there are now signs of reform, even on this side, in this most important part of rail-making.

As to the rules of the inspection proposed by Mr. Hunt, I observe that rails not coming fully up to the specification may be accepted as "No. 2." We have no such practice in Europe. Rails are accepted, if strictly up to specifications, or they are rejected and disposed of by the makers to contractors on less important roads, where low speeds are maintained.

In conclusion, the inspector working to this specification seems to me to be spared the unpleasant duty of rejecting any rails; for the bending-test can hardly fail to go through when repeated, and the rails not up to the strict terms of the specification can be passed as

"No. 2." The makers have every reason to be happy; but it remains to be seen whether the consumer also will be so.

3. The third cause of inferior rails now-a-days, as compared with those first rolled, namely, the *driving*, is perhaps the most important of the three, particularly because there is no help for it. To ask the maker to give more time to both the chemical and the mechanical processes, would be regarded as a suggestion to return to the old stage-coach. Nevertheless, the fact is, that no sooner is the pig-iron in the converter than it is out as a rail. Any one having the least experience in chemical processes must know that time cannot be dispensed with, but is necessary for dissolving, precipitating, and mixing, and for the due escape of the gases produced. Why is the Siemens-Martin steel more homogeneous than the Bessemer? Simply because time is given.

Again, in mechanical respects, the old mills, running at slow speed, produced a close grain in the rails, because the pressure was given in the stage of brown heat. This made the rails physically hard, as has been very clearly stated in a recent paper by Mr. Frederic A. Delano, of Chicago (*Trans.*, xvi. 594). This explains why different wearing-results might be got from the same chemical composition. One rail might be rolled at red heat, and the other at brown or nearly black. The thick flanges formerly prevalent permitted such cold-rolling without consequent rejections or waste. Since the thin flanges, like razors, have come into use, makers have been actually bound to spin them out in quick-rolling mills; and the only part that gets cold-rolling is the flange, while the head is always coming out red-hot. No close grain or physical hardness can be expected in the big heads—which is just the reverse of what should be the result sought.

However, it is of no use to ask makers to roll slow, at least not until engineers will return to thick flanges of, say, $\frac{5}{16}$ inch or rather $\frac{1}{3}$ inch; and even then, the ambition to turn out a rail a minute, and so many tons out of a converter, will not be given up. The only remedy I can now hope for is, that through the adoption of these thick flanges the rail-steel might with safety be made harder chemically, by admitting as much carbon (and silicon to give solidity) as the amount of phosphorus present will permit. It is in this view that my forthcoming paper closes with a suggestion of thick flanges and the use of steel base-plates, where the preservation of sleepers will necessitate a broader base than the rail-flange can practically give.

MR. HUNT (communication to the Secretary): I am glad so distinguished an authority as Mr. Sandberg agrees with me at so many points. In defense of my test-bars and bending-test, I would say that my specifications were drawn with a view of obtaining good and safe rails, but at the same time interfering as little as possible with the established practices of American mills. In several of these, the rails are loaded on the shipping-cars, both day and night, as they come from the drill-presses, at the rate of, say, a rail per minute; hence to hold the rails until a drop-test has been made would seriously interfere with the mill-practice. Moreover, steel is so sensitive to physical treatment that, to give the drop-test an absolute value, a piece should be taken from each rail. Of course, this cannot be done under existing circumstances. By my plan I secure samples from two different parts of each heat, and test them *before* the steel has reached the condition of a finished rail; and I know from years of experience, that, if properly conducted, this method is reliable. It is the inspector's duty to see that the test is properly conducted. Such supervision is equally necessary to the drop-test. I think I have known of railway engineers, looking wisely on while such tests were made, and drawing conclusions from them, when the results were controlled almost as much as though other steel had been tested. Now that I would accuse any one of presenting false results; but it is very embarrassing to have failures when one particularly desires success.

Accidents will happen, and I therefore permit a second test, but it must be from steel representing the same ingot. It is not intended that a *coaxing through* of the test-bar shall be allowed; but, as the rejection of ten tons of metal is a serious matter, I propose an honest protection of the maker's interest. If he seeks to obtain any more, the inspector is there for the purpose of calling a halt.

In relation to the matter of solid tops to the ingots, I fully agree with Mr. Sandberg that such ingots can be made; but we are not certain of always getting them. An ingot is a steel casting, and we are all familiar with the character of the sink-heads which the makers of steel castings have to provide. Cutting off the amount of steel specified by me need not necessarily increase the cost of rails. Such pieces can be used for other purposes, in which any such defects will play an unimportant part, and the crop-, or rail-ends, can be made so much shorter.

In regard to accepting unsatisfactory rails as No. 2, probably Mr. Sandberg is not aware that some American railroads accept a stipu-

ated percentage of such rails, while others do not. This is a matter, varying with each contract. When accepted it is usually at about 10 per cent. discount from the price of No. 1 rails, and seldom in quantity over 3 per cent. of the whole rolling. Hence inspectors under my specifications may have some "unpleasant duty."

As the railroads improve the character of their road-beds, they require more carefully finished rails; and conscientious inspectors will find plenty of work, if they perform their duty and examine *each* rail for line and surface defects, besides wanting a constant supervision of the section, measurement, sawing, chipping and filing, and drilling. In doing this, they will assist rather than antagonize the mill-management; for of course all makers desire to give their customers good rails.

NOTE ON THE CULTIVATION OF MUSHROOMS IN ABANDONED MINES AT AKRON, N. Y.

BY WILLIAM Y. WARREN, BUFFALO, N. Y.

(Read by invitation of the Council at the Buffalo Meeting, October, 1888.)

MESSRS. Thomas & Cross, having leased from the Akron Cement Company from twelve to fifteen acres of abandoned cement-tunnels and chambers, for the purpose of propagating mushrooms for the market, commenced operations about a year ago, and are now beginning to furnish mushrooms in quantities to the local markets of Buffalo and Rochester.

The quarries, or, more properly, the mines, of the Cement Company are located at Akron, Erie County, N. Y. The cement-rock occurs in a stratum about 7½ to 8 feet in thickness, which is overlain by from 15 to 25 feet of limestone and flint-rock. There is a gentle rise from the opening of the tunnel inwards, giving a natural drainage and leaving the chambers comparatively dry. The roof is good and is supported by pillars and heavy timbers.

The mushroom-beds are located about 300 feet from the opening of the tunnel, and are made in the form of mounds about 3 feet by 12 feet, at the base, and 2½ feet high, giving to the chambers the appearance of a subterranean graveyard. Many experiments have been made to determine the best form for the beds, and the mound

is now deemed superior to any other, giving, as it does, a much larger producing-surface in proportion to the area occupied. The beds are constructed of fresh horse-manure and garden-loam, in the following manner:

Frames, 3 by 12 feet in size, made of 2-inch by 4-inch scantlings, are laid in a row upon the floor of the chamber, about 8 to 12 inches apart, leaving room to pass between them. Upon each of them a mound of the fresh manure, which has been well forked over, is made with sloping sides, about 2½ feet high. This is pressed down firmly, and the mushroom spawn is planted upon the entire surface, which is then banked up and completely covered with from 2 to 2½ inches of carefully selected black garden-loam.

A bed will commence to produce in small quantities in from ten days to two weeks, and will last from ten months to a year, when the strength of the manure will have become exhausted, and the spawn (which occurs in the form of thin white fibers like delicate roots) must be replanted in fresh beds.

The form of the beds has not been the only subject of experiment in the enterprise. Their location, also, has had careful consideration and required many practical trials. There is now in this locality about half an acre covered with the beds. On the day of our visit twenty pounds of mushrooms had been gathered, and the product was expected to be much larger a month later, from new beds which were soon to commence bearing.

The temperature of the tunnel is nearly uniform the year round, running from 50° to 55° F. No artificial heat is used, and the beds are not enclosed in any way, from the entrance or the rest of the tunnel. The atmosphere is naturally very damp. During the summer, when the wind was in a favorable direction to blow into the tunnel, and the temperature was raised several degrees, it was noticed that the mushrooms grew much more rapidly. If the temperature were artificially raised and the chambers kept at a uniform heat the product would probably be largely increased.

No artificial light is employed except such as is necessary for the men in making new beds or picking the crop, for which purposes they use large gasoline torches.

The beds require no watering or further attention, until they commence to give out and require replanting.

A similar enterprise has been very successfully carried on at Utica, Ill., from which place from 1000 to 1500 pounds of mushrooms per week are sent to the Chicago market and find a ready sale.

The projectors expect to be able to raise pie-plant and possibly celery, lettuce, and other vegetables, under somewhat different conditions as to light and heat, which they propose to investigate as soon as their mushroom-beds are successfully in operation on the scale contemplated.

CEMENT-ROCK AND GYPSUM-DEPOSITS IN BUFFALO.

BY JULIUS POHLMAN, BUFFALO, N. Y.

(Buffalo Meeting, October, 1888.)

CONSIDERING the truly wonderful natural resources of the United States, and the variety and extent of uses for hydraulic cement at the present day, it seems strange that the manufacture of the artificial product called Portland cement has not been introduced here. Possibly the reason may be we have in different States of the Union rock deposits from which a cement can be obtained that is as good, and some claim even better, because of a more uniform quality, than the English product after it has been subjected to transportation across the ocean.

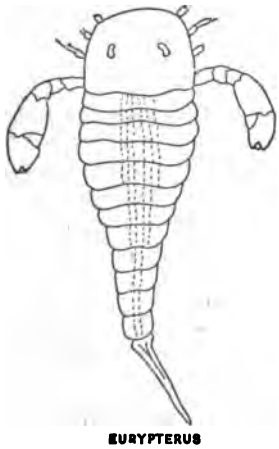
One of these natural cement-rock deposits is found in the northern part of the city of Buffalo, and its manufacture has been carried on successfully for the past eleven years by the Buffalo Cement Co., Mr. L. J. Bennett, President. The quarries cover an area of about two hundred acres, of which only twenty-five acres have been exhausted up to date; and the supply is assured for years to come. One acre yields on an average 80,000 barrels of cement; hence 12,000,000 barrels is a low estimate of what these quarries will yield before they are exhausted.

According to the Geology of New York this cement-rock deposit is called the *Water-Lime group*, and forms in this section the uppermost member of the Upper Silurian. Its outcrop can be traced along the base of the second limestone terrace of Western New York, but at Buffalo a very large part of the overlying Corniferous limestone has been removed by natural agencies, leaving only about eight feet (six feet of rock and two feet of clay) on the top of the cement-rock, and thus furnishing the conditions for successful quarrying at a minimum expense.

This rock is really an impure limestone, a mixture, as it were, of the underlying shales of the Onondaga Salt group with the overlying

Devonian limestone. It is about five feet thick, has a peculiar grayish-bluish tint, and emits a metallic, ringing sound when struck with a hammer. It can easily be recognized by these characteristics, but better yet by its peculiar fossils, *Eurypterus* (see Fig. 1), *Pterygotus* and allied genera, which appear to have inhabited the waters to

Fig. 1.



EURYPTERUS

the exclusion of any other form of animal life during the time the rock was deposited.

A series of comparative analyses of different brands of cement seems to demonstrate that the high-priced artificial English product is not needed here. Professor E. T. Cox, former State Geologist of Indiana, gives his results as follows:

	"K. B. & S." (English Portland.)	"B. C. Co." (Buffalo Portland.)
Silicic acid,	31.43	32.86
Alumina,	10.80	10.40
Lime and magnesia,	57.77	56.74
	100.00	100.00

Professor J. J. Brown, of Syracuse University, gives the composition of the Buffalo brand as follows:

Silicic acid,	32.22
Alumina,	10.91
Lime and magnesia,	56.75
Iron,	.12
	100.00

The cement made at Buffalo has been used in many of the most important works in the city and throughout the country, and a number of the leading railroads have adopted it as their standard brand.

While drilling for water a few years ago, near the kilns on the premises, Mr. Bennett conceived the idea of making a complete record of the rock-section by using the diamond drill. The following core was obtained, beginning where the five-foot layer of cement-rock had been removed.

Waterlime Group.	1– 25 feet, shale and cement-rock in thin streaks;	
	25– 30 feet, tolerably pure cement-rock;	
	30– 43 feet, shale and cement-rock in thin streaks;	
Onondaga Salt Group.	43– 47 feet, pure white gypsum;	
	47– 49 feet, shale;	
	49– 61 feet, white gypsum;	
	61– 62 feet, shale;	
	62– 66 feet, white gypsum;	
	66– 73 feet, shale and gypsum, mottled;	
	73–131 feet, drab-colored shale, with several thin layers of white gypsum;	
	131–133 feet, dark-colored limestone;	
	133–137 feet, shale and limestone;	
	137–140 feet, compact shale;	
	140–720 feet, gypsum and shale mottled and in streaks;	
	720–725 feet, limestone.	

The drill continued to the depth of 1305 feet, finding only soft red shale, with a solid band of white sandstone, between 760 and 785 feet.

Permanent water was found at a depth of 43 feet running through the first layer of white gypsum; at 452 feet gas was struck in fair quantity and good quality, which has since been used in connection with that obtained from three other wells, sunk near by, for the generation of steam in the cement works. Salt water was found at 555 feet, which left on evaporation about 12 per cent. of salt.

This core demonstrates that Buffalo is underlain, at the depth of about 50 feet, by a deposit more than 20 feet thick of white gypsum. For the purpose of determining the advisability of mining this deposit the Cement Company had a shaft sunk, 20 feet square; but the rush of water through the upper layer of gypsum (43 to 47 feet) effectually stopped the progress of the work. A steam-pump, with a capacity of 2000 gallons per minute, did not make any

impression upon it, and the theoretically correct, but practically impossible enterprise of gypsum mining in the city of Buffalo was abandoned.

The spot where the drilling commenced is 70 feet above the level of Lake Erie (573 feet above the ocean); hence the waters running along the gypsum-layers must come from the higher lands to the east and south of the city, and not from the lake.

FERRO-SILICON, AND THE ECONOMY OF ITS USE.

BY W. J. KEEP, C.E., DETROIT, MICH., AND EDWARD ORTON, JR., E.M., COLUMBUS, OHIO.

(Buffalo Meeting, October, 1888.)

DURING the past two or three years consumers of pig-iron have been seeking more knowledge regarding the chemical questions involved in foundry-practice. This desire has been increased by the papers of Professor Thomas Turner, of Mason College, Birmingham, reporting a series of tests made to show that silicon is a useful, rather than a damaging element when present in proper quantities in cast iron, and that by its use pig-irons and scrap, which when used alone are totally unfit for foundry purposes, may be made to give satisfactory results. Professor Turner has shown that the strength of a cast-iron depends on :

1. The amount of weakening impurities present;

2. The proportion existing between the combined and the graphitic carbon in the cast-iron.

He says that the tendency of combined carbon is to increase hardness and brittleness, while graphitic carbon makes iron soft, malleable and tough; too much of either form is a disadvantage; the strength depends on the proportion.

Strength being the thing most desired, irons having an excess of weakening impurities will not find a market, and therefore the only thing to provide for is the proper proportion between combined and graphitic carbon. Professor Turner shows that by a judicious use of silicon this proportioning can be accomplished exactly according to the wish of the founder; an increase of silicon changing combined carbon to graphitic, and *vice versa*. When the founder understands its use he may soften and toughen, or harden and strengthen his

iron to suit his requirements; but Professor Turner warns him against the use of silicon, without first understanding when it is needed; for in an iron where the carbon is already graphitic, more silicon may weaken it and make it brittle.

From the effects produced by the extensive use of high-silicon irons for many years, and from the discussion of the subject during the past few years, silicon has come to be considered as a softening agent.

In the United States for a number of years past many foundrymen have depended upon the irons made from the lean ores of Ohio and Kentucky as softeners to counteract the hardness of irons made from the refractory ores of Lake Superior and northern New York and the irons of the South; but it has only lately been generally understood that this softening quality is due to silicon.

Upon examining the analyses of Ohio softeners made from native ores, it will be found that the percentage of silicon runs from 3 to 7 per cent., and that none regularly made previous to 1887 contained more than 7 per cent.

As the process by which silicon accomplishes these desirable results has become better understood, the demand for high-silicon iron has increased. In 1887 foreign irons containing as high as 10 per cent. of silicon were imported into the United States. These high-silicon irons go under the name of ferro-silicon. As the call for this imported product increased, ferro-silicon containing from 7 to 14 per cent. of silicon, and of a very high grade in other respects, has been produced in America to supply this demand.

The question has naturally arisen, whether the use of irons containing such a high percentage of silicon is as economical as when the percentage is lower. This has led us to the experiments and conclusions set forth in the present paper. We shall treat the subject under the following general divisions: first, a comparison of the composition of foreign and American ferro-silicons; secondly, the inquiry, whether the silicon in the pig is retained in the iron when remelted; and thirdly, the inquiry, whether the silicon in the pig is imparted to other irons during remelting, without loss of silicon.

I. *A Comparison of the Composition of Foreign and American Ferro-Silicons.*

To make this comparison we have selected from the stock of irons in our possession, and with which we are familiar, having repeatedly tested them, the following examples:

"Govan" ferro-silicon, from a prominent Montreal firm, the analysis being published by them. (Marked *a*, in Table I.)

A foreign ferro-silicon, obtained from a well-known New York broker, and analyzed for our tests by Fleming. (221 in Table I.)

The ferro-silicon used by Professor Turner in making his tests, the analysis being published by him. (This iron we have not ourselves tested. It is marked *b*, in Table I.)

The only American ferro-silicon with silicon as high as 10 per cent. that we have heard of as being made regularly, is the "Pencost" brand, the producers of which kindly furnished us with all the metal that we needed for our experiments.

Nos. 403 and 401 of the table, representing American ferro-silicon, are of this iron. These irons were analyzed for our tests by Mr. Orton and Professor Lord.

We add, for purposes of further comparison, four analyses of two well-known brands of Ohio softeners, namely, "Wellston" (*c* and *d*, analyzed by Britton) and "Globe" (No. 178, analyzed by Setterwall, and *e*, analyzed by Orton).

We could add analyses of other high-silicon irons, both foreign and American, that we have in our possession; but for the present purpose these will suffice. The following table gives the analyses referred to.

TABLE I.

	No. of Test.	Kind of Iron.	Silicon.	Combined Carbon.	Graphitic Carbon.	Manganese.	Phosphorus.	Sulphur.
Ferro-Silicons.	a	Foreign, "Govan".....	10.55	1 84	.52	3.86	.04	.03
	221	" (New York broker)	10.62	...	...	2.32	...	...
	b	" used by Prof. Turner	9.80	.69	1.12	1.95	.21	.04
	403	American, "Pencost".....	12.08	.06	1.52	.76	.48	tr.
	401	" "	10.34	.07	1.92	.52	.45	tr.
Softeners.	c	" "Wellston".....	6.67	...	2.57	...	.50	tr.
	d	" "	5.06				.75	.05
	178	" "Globe".....	5.89	.30	2.85	1.00	1.10	.02
	e	" "	6.64				.99	tr.

The points which attract attention in this table are:

1. The large proportion of combined carbon in the foreign irons. If this carbon remains in the combined state after the ferro-silicon is added to the foundry mixture, it will tend to harden the resultant casting. The carbon of the American ferro-silicon is in the graphitic state, to begin with, and therefore such irons should be the superior softeners.

2. Manganese in the foreign irons runs much higher than in the American. It is the general belief that manganese acts more powerfully than silicon and in an opposite direction, causing carbon to assume the combined state. If this is the case, a large part of the silicon in the foreign irons would be needed to neutralize the effect of their contained manganese, leaving much less silicon to exert a softening influence upon the combined carbon of the hard iron which it is expected to soften, than would be the case if the manganese were absent.

In the American silicon-irons the manganese is so low that the silicon is free to act directly on the combined carbon of the iron to which it is added.

3. The foreign irons are nearly free from phosphorus, while "Pencost" contains about one-half of one per cent. and the other American irons which we have mentioned contain about one per cent.

We may remark that in most foundry mixtures phosphorus will not run below one-half of one per cent., and therefore the effect of the phosphorus, added by the use of American ferro-silicon, will not affect the mixture. Such a small quantity of these high-silicon irons is needed to produce desired results, that if they contained no phosphorus whatever this use could not reduce phosphorus in the mixture one-tenth below its former amount.

Again, perhaps a small percentage of phosphorus in foundry-iron may do more good than harm.

4. We observe the high percentage of silicon which these irons, both foreign and American, contain, as compared with the American softeners made in former years. This increase in silicon percentage has led to the name ferro-silicon.

II. *Is the Silicon in the Pig Retained in the Iron when Remelted?*

The high-silicon irons are made in a furnace at a very high temperature, and it has been asserted that when remelted in the foundry, at a much lower temperature, the iron will drop much of its silicon, which will be carried off in the slag.

The introduction of these high-silicon irons has led to a discussion as to the relative value of softeners, with high or low silicon, and also as to the relative economy in the use of a 10 per cent. metal, as compared with an iron in which the silicon reaches only half that percentage. We wish it distinctly understood that we discuss in this paper, not the question, whether the silicon is oxidized by the blast of the foundry cupola, or not, or how much may be lost in that way; but simply whether remelting at a lower temperature will release a portion of the silicon, and if so, in which iron, that is, high- or low-silicon iron, the loss is the less. We have made a number of tests to arrive at the truth.

We secured seven pigs of "Pencost" ferro-silicon, containing silicon ranging from 4¼ to 12 per cent. These pigs (except the first one) were made at the same furnace and from the same stock, and under the same general conditions, which would insure a similar composition in each, except as to the silicon, and the change in carbon which the variation in silicon would produce.

Sets of test-bars were made from each of these pigs and tested by "Keep's Tests;" the crucible-temperature being about the same as that of the foundry-cupola. These test-bars were then again remelted and a second series of bars was cast and tested. Both the first melts and the remelts were then analyzed, with the results shown in the following table:

TABLE II.

Silicon in Original Pig of Pencost Ferro Silicon.	No. of Test.	Silicon in Test Bars, 1st Melt.	No. of Test.	Silicon in Test Bars, 2d Melt.	Actual loss of Silicon from 1st Melt.	Actual loss of Silicon from 2d Melt.	Percentage loss from 1st Melt.	Percentage loss from 2d Melt.
4.36	397	4.35	405	4.25	.01	.10	.22%	2.52%
6.76	398	6.75	406	6.57	.01	.18	.14%	2.66%
8.09	399	8.07	407	7.99	.02	.08	.24%	.99%
9.42	400	9.27	408	9.31	.15	gain	1.59%	...
10.34	401	10.29	409	10.24	.05	.05	.48%	.48%
11.34	402	10.83	410	10.76	.51	.07	4.58%	.64%
12.08	403	11.68	411	11.48	.40	.20	3.31%	1.71%

An examination of these results reveals that on the first melting the irons having less than 10 per cent. of silicon lost on an average

0.55 per cent., while those containing 10 per cent. and over lost 2.80 per cent. of the silicon held in the pig.

On the second melting the losses are twice as great in the low as in the high percentages. From these observations we conclude that though the absolute loss is greater in the higher numbers, it is still so small in all cases as to be unworthy of consideration.

To further prove the question, whether irons high in silicon can hold their silicon when melted at lower temperature, a series of pigs, from an Ohio furnace now out of blast, was tested, and the pigs and bars analyzed. These pigs were all made from the same mixture of ores and represent all the grades of iron made at the furnace. The results appear in the following table:

TABLE III.

No. of Test.	Grade of Iron.	Silicon in Original Pig.	Silicon in 1st Melt.	Loss of Silicon.
35	Silvery flaky	6.99	6.92	.07
34	" "	6.86	6.80	.06
33	" "	6.82	6.75	.07
36	" (open)	5.66	5.63	.03
39	A. 1 Foundry	5.15	5.20	+.05
37	B. 1 (Hot)	4.66	...	...
38	B. 1 (Cold)	4.05	4.00	.05
40	No. 2 Foundry	3.64	3.44	.20
41	No. 1 Mill	3.57	3.59	+.02
42	White	1.97	1.98	+.01

These tests show great uniformity in the losses of silicon, which are in all cases trifling.

A careful study of these two series of tests of Ohio silicon-irons, the silicon ranging from 2 to 7 per cent. in the ordinary softeners, and from 5 to 12 per cent. in the "Pencost" ferro-silicons, or, taking both tables together, from the white iron, made so by low silicon, to the whitish iron produced by an excess of silicon, leads us to conclude:

1. That silicon pig-iron loses practically none of its silicon in remelting, through being melted at a lower temperature.

2. That there is practically the same actual (not proportional) loss from irons, with comparatively low silicon, as with high, and

therefore one is as economical as the other, so far as the silicon is concerned.

Before leaving the question of loss in silicon, due to the lower temperature in remelting, let us compare the losses in foreign with the losses in American ferro-silicons. In the following table we exhibit the determinations of silicon in a 16 per cent. ferro-silicon (396), imported by a prominent New York broker; a 12 per cent. metal (213), imported by the same firm; and a 10 per cent. metal (214), imported by a Montreal broker.

These samples are sufficient to represent foreign irons, and we introduce Nos. (403) and (401), as representing American ferro-silicons.

TABLE IV.

Number of Test.	Kind of Iron.	Silicon in Original Pig.	Silicon in First Melt.	Silicon in Second Melt.	Percentage loss in First Melt.	Percentage loss in Second Melt.
396	Foreign (New York Broker).....	16.32	14 47		11.33%	
213	" " " "	11.99	10.84		9.60%	
214	" "Govan".....	9.85	9.81		.40%	
403	American, "Pencost".....	12.08	11.68	11.48	3.31%	1.71%
401	" "	10.34	10.29	10.24	.48%	.48%

These results show distinctly that these foreign irons lose in remelting more than the American irons with which they are compared. We will not endeavor to account for this in the present paper.

III. *Is the Silicon in the Pig Imparted to other Irons without Loss?*

That is, will ferro-silicon hold its silicon until mixed with irons deficient in silicon; and, for making such mixtures, will high or low-silicon iron lose most silicon?

To arrive at conclusions regarding these questions, we made four series of melts. Each series was composed of four casts and each cast of six sets of test-bars. We used for these a white-iron base (376), with 0.186 per cent. silicon and a gray-iron base (253), with 1.249 per cent. silicon; and for adding silicon we used the first pig of "Pencost" (397), before spoken of, containing 4.36 per cent. of silicon, and the fifth pig (401), containing 10.34 per cent. silicon.

The first series was white (376) and 4.36 per cent. of ferro-silicon (397).

The second series was white (376) and 10.34 per cent. of ferro-silicon (401).

The third series was gray "FLM." (253) and 4.36 per cent. of ferro-silicon (397).

The fourth series was gray "FLM." (253) and 10.34 per cent. of ferro-silicon (401).

The four numbers of each series were to contain respectively 1.5, 2, 2.5 and 3 per cent. of silicon. Having been weighed with great care, they were melted so as to avoid oxidation, and were cast into bars and tested, when each set of bars was analyzed, with the results shown in the following table:

TABLE V.

Percentage of Si. calculated to be in Test-Bars.	Number of Test.	Silicon found in Bars White (376) and 4.36 per cent. Pencost (397).	Number of Test.	Si. found in Bars White (376) and 10.34 per cent. Pencost (401).	Number of Test.	Silicon found in Bars Gray FLM (253) and 4.36 per cent. Pencost (397).	Number of Test.	Silicon found in Bars Gray FLM (253) and 10.34 per cent. Pencost (401).	Losses of Silicon in White (376) and 4.36 per cent. Pencost (397).	Losses of Silicon in White (376) and 10.34 per cent. Pencost (401).	Losses in Silicon Gray FLM (253) and 4.36 per cent. Pencost (397).	Losses of Silicon in Gray FLM (253) and 10.34 per cent. Pencost (401).
1.50	340	1.49	344	1.48	348	1.50	352	1.53	—.01	—.02	±.0	+.03
2.00	341	1.97	345	1.92	349	2.01	353	2.11	—.03	—.08	+.01	+.11
2.50	342	2.55	346	2.42	350	2.42	354	2.41	+.05	—.08	—.08	—.09
3.00	343	2.95	347	2.87	351	2.86	355	2.86	—.05	—.13	—.14	—.14
Average deviation from calculated per cent.....									.03	.08	.06	.09

A careful examination of this table reveals the following facts:

1. Losses in silicon increase as the percentage of silicon in the mixture increases. That is, an iron deficient in silicon can have its silicon raised to 2 per cent. with less loss than if raised to 3 per cent.

2. Losses are slightly higher when using the 10 per cent. than with the 4 per cent. iron; but the losses are so small in all cases that no commercial discrimination can be drawn in favor of one class over another in the matter of economy of silicon. In the extreme case of melting a white iron and a 10 per cent. silicon metal together, to produce a dark soft foundry-iron, there was no excessive loss in silicon.

IV. *General Conclusions.*

We may draw from the results presented the following general conclusions:

1. Pig-irons of any grade may be melted and cast without excessive loss in silicon or excessive change in chemical composition.

2. Ferro-silicons lose in remelting a little more silicon than do the softeners containing less silicon; but the difference is so small that commercial distinction is not possible.

3. In alloying silicon-irons with scrap or pig-iron deficient in silicon, the silicon is practically all retained in the mixture, and no economy in silicon results from the use of one class of American silicon-iron over another.

4. American ferro-silicon, so far as we have investigated, is more economical than the imported, and probably, in most cases, softens to a greater extent.

We think that we have accomplished what we set out to do. Our discussion is altogether from a chemical standpoint, and we have proved that the founder can produce a desired grade of casting by compounding suitable irons, as cheaply, if not more so, than by purchasing irons in which the combination is already what he desires.

As soon as we can prepare a report upon the physical characteristics of these irons and the mixtures that we have made, we shall publish such results.

HOT-SPRING FORMATIONS IN RED MOUNTAIN DISTRICT, COLORADO: A REPLY TO THE CRITICISMS OF MR. EMMONS.

BY THEODORE B. COMSTOCK, CHAMPAIGN, ILL.

(Buffalo Meeting, October, 1888.)

THE paper of Mr. S. F. Emmons on the structural relations of ore-deposits, presented at the Boston meeting in February last, having been, as I am informed, unavoidably delayed in publication, was not distributed in the preliminary edition to members until immediately before the sheets of Volume XVI. of the *Transactions*, containing it, were sent to press. Consequently, I saw it for the first time too late to offer any comments which might have accompanied it in that volume. This is my justification for present-

ing as a separate contribution to a subsequent volume what would have been more appropriately placed as a part of the discussion of Mr. Emmons's paper, and also for repeating here, from Volumes XV. and XVI. the passages necessary to explain the point at issue.

In a paper on the geology and vein-structure of southwestern Colorado, I used the following words (*Trans.* xv., 260) concerning the Red Mountain district, in that region:

"The district is peculiarly the area of secondary action, the *locus par excellence* of the hot springs. Either this section was well protected during the Glacial period, or the ore-deposits which have made the district famous are of post-Glacial date. There is every reason to adopt the latter view, and this will explain much of what would otherwise seem puzzling in the present drainage. It would be presumptuous for me to controvert Messrs. Becker and Emmons upon their own ground; but I strongly suspect that they have worked too much in fields where only the latest phases of vein-formation are well expressed. I have now examined nearly every prominent mining district between latitude 37° N. and latitude 49° N., east of the meridian of 112°. In all that territory I have had no evidence of the local segregation of ore-deposits where one of the prime conditions was not the prior existence of a vein or other special deposit containing the metal. Every requisite except this has existed in many places in the west, in our district as well as northward, yet there are numerous instances in which no such deposits were formed. All over the central San Juan area there are the relics of hot springs, some of which are active to-day. In not a few of these, but always where veins occur, ore-masses have been segregated; but many of them which traversed the same country-rock at the same period have left no metalliferous deposits except the iron oxide which is almost universal.

"Along the line of the axes of the Arsenical and Antimonial zones previously mentioned, for the distance of fifteen miles, Red Peak lying about midway, the hot spring formations are abundantly exposed. The area of greatest activity was in what is now the Red Mountain mining district. For about three miles along the upper end of Red creek, a belt not more than three miles in extreme width, but extending across the divide towards Cement creek, thence south along the Red Mountains to the junction of North Mineral and South Mineral creeks and beyond, there is a belt of extinct hot springs which rivaled those now existing in the Yellowstone Park. The ore-deposits in the heart of this mining district are commonly in mounds or ridges of siliceous sinter, and very many of them are extinct geyser-chimneys. No one can rightly interpret the structure of this area without some familiarity with such phenomena as are to be witnessed to-day in the Park."

In his recent paper, Mr. Emmons comments (*Trans.* xvi., 809) upon this passage as follows:

"Owing to the vagueness of Mr. Comstock's statements, and his many misconceptions of geological facts, we cannot always be sure of his meaning; but from a careful consideration of this paper I infer that he considers the head of Red Mountain creek in that region, which he calls a crater-like depression, to be an extinct geyser-basin, and that the ore-deposits opened by the mines now worked there have been formed by geysers. Now, a geyser, as its name implies, is a spring which

violently ejects its waters into the atmosphere, and hence, even more than other springs, is essentially a surface-phenomenon.

"As, since the ore-deposits of Red Mountain basin were formed, something like 2000 feet in thickness of rock-material have been eroded away to form the basin, it is evident that they could not have been formed by a geyser, nor can it be determined at the present day whether geysers ever existed there or not. Apparently, Professor Comstock has thought to find a resemblance between certain rounded ridges of light-colored rock near these mines with the mounds formed by deposits from the springs in the Yellowstone Park. Instead, however, of being surface deposits, they are simply portions of the andesitic country-rock from which acid waters have removed the basic constituents, perhaps depositing a certain amount of silica in their place; the resulting quartzose masses offering greater resistance to the disintegrating effect of atmospheric agents and to erosion than the surrounding rocks, have been left as mound-like ridges protruding above the general surface, more or less independent of the natural drainage-channels.

"There are abundant mounds formed by deposits from springs in the vicinity of Red Mountain, but they consist mainly of bog-iron, and are formed by oxidizing surface-waters passing through and decomposing bodies of sulphides of the metals, whose action is quite distinct from, and in some sense the reverse of those which originally formed these bodies."

I am not aware of any greater degree of "vagueness" in my statements than was necessitated by the wide range of observations generalized in my paper. It was impossible to enter into all the details, even though, by so doing, I might have forestalled some mistaken censure. As to my "many misconceptions of geological facts" (none of which are specified, unless what appears above is intended as a specification), I will not affect to consider the phrase intentionally offensive. Mr. Emmons has more than once characterized in this tone the views of abler men than myself, who differed with him upon subjects connected with vein-formation; and I am accustomed to consider it as nothing more than his way of describing an interpretation of geological facts which does not agree with his own. It is not worth while to resent a mere idiosyncrasy.

But Mr. Emmons is not warranted by anything in my mention of the Red Mountain district in inferring that I suppose the deeper ore-deposits opened by the mines now worked there to have been formed by geysers. I had observed the fact that certain mounds in the Red Mountain basin have such structural peculiarities (not surface-contours merely) as indicate an origin similar to that of the geyser-mounds of the Yellowstone Park. In stating my conclusions that many of these mounds or ridges of siliceous sinter are extinct geyser-chimneys, I did not express the idea that the *original* formation of the ore-bodies which do undoubtedly occur in them, was due to geyser-action. On the contrary, as the context clearly shows, I be-

lieve the indications are that only the so-called secondary, or oxidized deposits, were the products of that stage.

Mr. Emmons has detected, in his rapid survey, what is very familiar to me also, namely, the "rounded ridges" resulting from disintegration and erosion; and he infers without reason that my views are based upon a mistaken notion as to the character of these forms of relief. As my generalization rests upon other grounds entirely, his exposure of my supposed error is not pertinent.

It should be added that many of the mines now worked in this district have been opened since my paper was written, in 1886. The Yankee Girl (the only one named by Mr. Emmons in his criticism) is in a different part of the basin, and presents none of the surface-features to which my statement makes allusion. Moreover, in the same paper (*Trans.* xv., 261) it was remarked that "the Yankee Girl, Guston, Congress, etc., appear to occupy the bowls of non-sintering springs," and no claim was put forward that any of them are in geyser-mounds. When the paper was written very little development had been made at those points, and no very definite statement could be uttered concerning them.

It is quite possible that the views I have expressed may be modified or disproved by the progress of mining developments and more thorough geological field-work. But such as they are, they express the results of a study of the rock-structure over many square miles of territory, and not of the whimsical or superficial reasoning which my critic assumes. The structural geology of that region is only beginning to be unravelled; and Mr. Emmons, who has won such deserved credit by his faithful and intelligent labors in other parts of Colorado, should not make haste to condemn, before he has even comprehended, the observations and deductions of other conscientious workers. The conclusions reached by Mr. Emmons in his recent paper are, so far as they affect the same areas, equivalent to the view expressed by myself early in 1886, as a comparison of the two articles will show.

THE MINING INDUSTRY IN ITS RELATION TO FORESTRY.

BY B. E. FERNOW, CHIEF OF FORESTRY DIVISION, WASHINGTON, D. C.

(Buffalo Meeting, October, 1888.)

IN order to ascertain to what extent the mining industry has been dependent upon forest-supplies, for the purpose of a report upon

the relation of mining to forestry, Mr. F. P. Dewey, curator at the National Museum, undertook at my instance to canvass the several mining-districts. As is usual with such inquiries, the result does not compare with the effort, and one is led to mourn the absence of general interest in an important economic question, even among so enlightened a body of citizens as the Mining Engineers.

If an appeal to the members of the Institute could be expected to produce a reform of their ways in this particular, I would call upon them to send us voluntarily the information they have on mine-timbering, a subject which has found but little recognition in the literature of mining in this country, and concerning which a compilation of usages and experiences in different localities would form a desirable contribution, at least to the history of mining. It may be sad, as the poet assures us, to reflect upon what might have been; but it is wholesome to know what has been and is, and to consider what will be,—or even, now and then, what ought to be.

The questions of main importance in the above-mentioned inquiry were comprised in the circular shown on the following page.

Of the 3,000 circulars sent, about 20 per cent. received replies, and of these but one-half have been found of value.

From the data thus obtained it is evident, that only a rough estimate can be made as to the probable consumption of timber in mining operations. The uncertainty of figuring upon so scanty a basis is of course increased by the fact that the use of timber in mines varies beyond computation even in the different mines of the same region, according to the condition of the walls, drainage, depth, kinds of timber, etc. It is merely in order not to be without some figures in these days of statistics, that I have attempted to compute the probable annual consumption for mine-timbering.

The result thus tentatively reached is, in round numbers, 150,000,000 cubic feet of wood per annum.

Of this consumption we can with some degree of certainty claim 35,000,000 cubic feet of mine-timber for the anthracite mines, being at the rate of about one cubic foot per ton. This estimate is based upon the actual consumption of the Philadelphia and Reading Coal and Iron Company, mining over 15 per cent. of the total out-put. Figuring upon the same basis, from six to seven million cubic feet of sawed lumber would be needed outside the mines. Other reports representing another 15 per cent. of the out-put, give the consumption at something less than one-third of a cubic foot per ton.

The bituminous coal-mines use less timber per ton, but produce

CIRCULAR OF INQUIRY.

a. Give the region covered by your answers (whether a mining district, a group of mines, several isolated mines, or a single mine).

b. Give area of timber owned or controlled by users.

c. What is the condition and outlook of home supply of timber.

d. If any systematic plan of cutting timber or providing for future supply is followed, describe it.

1. TIMBER.

Amount used, (in 1,000 feet, B. M.)	Previous to 1880.	1880.	1881.	1882.	1883.	1884.	1885.	1886.
Total								
Above ground								
Under ground								
For new work								
For renewals								
Value at mine								
Where obtained								
Distance transported								
Kinds, with relative proportion of each								

2. TIMBERING.

Give methods of setting timbers, with sketches and character of ground in which each method is used.

Give average life of different timbers in the mine. .....

.....

Sizes most used. .....

.....

3. MINE.

When discovered? ..... When opened? .....

Distance from railroad or water transportation? .....

Character of country? .....

In what condition is worked out ground left within the mine? .....

PRODUCT.	Previous to 1880.	1880.	1881.	1882.	1883.	1884.	1885.	1886.
Character								
Gross value								
Gross amount								
WALLS.								
Hanging								
Foot								
Depth attained								

4. GENERAL REMARKS.

more coal. Figuring at the rate of one-half of a cubic foot per ton, which seems to be the average in the Cumberland region (the reports from other fields being lower, and from Westmoreland higher —1½ cubic feet), the total consumption of mine-timber would about equal that in the anthracite-mines; and, adding a similar quota for outside use, the total consumption of timber for coal-mining may be safely placed in round numbers at 85,000,000 cubic feet annually, the continual supply of which would require a forest-reserve of not less than 2,000,000 acres of well-stocked forest.

The use of timber in iron-mines varies considerably. While some of the Lake Superior mines seem to require over 2 cubic feet per ton, the open pits of other districts make little demands for timber. An estimate of 7,500,000 cubic feet for the 7,500,000 to 8,000,000 tons of iron-ore mined may appear reasonable.

In regard to precious metal and other mines, the basis for any kind of estimate is absolutely wanting: the data are too few. While the Bodie mine, for instance, used 4½ cubic feet per ton of ore, the Ontario could mine 100 tons with the average expenditure of 6¼ cubic feet. On the other hand, again, the consumption in such mines as those on the Comstock lode and at Leadville, where large ore-bodies are stoped, and the Nevada system of timbering and cribbing is employed, is enormous.

I think it safe to say that the estimate of a total annual consumption for mining purposes in the United States of 150,000,000 cubic feet of wood, which is here simply set down as the first round guess in this direction, is considerably below the truth.

In comparison with the general consumption of wood in the United States, which must amount annually to over 20 billion cubic feet, these figures do not allow us to point out the mining industry *in general* as a leading factor of the exhaustion of our timber-supplies. Yet in some regions the question of the supply of mining-timber has already attained sufficient importance to call for its consideration.

In the Pennsylvania coal-region the near-by supply is rapidly diminishing, making longer haulage necessary. In fact, the companies and operators reporting to the Forestry Division draw their supplies mostly 30 to 40 miles, and some of them 80 miles, to their fields of operation. Even from the famous timber-region of northern Michigan, this state of affairs is reported. In Utah, the scanty home supply would have been used up long ago, but for some importation of California timber, upon which also the Nevada mines

now rely almost entirely. The Montana mines are using enormous quantities, much more than can be furnished for any length of time by the limited home supplies. In Butte City, the great Anaconda mine alone consumed last year over 1,000,000 cubic feet of mining- and building-timber; and the consumption of firewood for smelting, etc., in this one camp, was estimated at a round 300,000 cords per annum.

While to the mine-superintendent other questions of management may seem more important at present, the question of timbering with regard to economy in the use of the material may soon become momentous, as the general lumber-market is bound to advance in price with increased demand and decreased supply, for the reproduction of which nothing but nature's slow processes are relied upon.

The low-grade ores of Utah, which are awaiting a comparative exhaustion of their richer neighbors, may have to wait longer for their day of prosperity, because the days of cheap timber are passing; and the competition in coal-mining may make the closest economy in every direction essential, just as it has become a necessity for makers of charcoal-iron to study economical processes of charcoal-production, in order to compete with other manufactures or with localities which have not yet exhausted or reduced their easily-obtainable wood-supplies.

The typical Western man, wholly occupied in building up the present, and accustomed to shift along as best he may with the means at hand, gives little thought to future needs, confident that some way will be found—whatever that may be—to supply them. Thus, his first suggestion, when shown the possible exhaustion of timber-supplies, would be that we might perhaps do without timbering; that metal structures may in future furnish a cheap substitute, at least in part, and in permanent situations; or that caving-in systems, or rock-filling, may prove capable of more extensive application. The latter system, which has recently been discussed in several papers before the Institute, and is used to more or less extent in the Michigan iron-mines and elsewhere, certainly recommends itself in respect of safety, permanence and facility for exhaustive mining, and on account of the inexhaustible supply of the filling material it employs. Even in the Menominee range, once one of the best-timbered sections of the country, this method seems to have proved itself almost 50 per cent. cheaper than the use of timbering.

The discussion of the possibility of extending the use of these systems, or other substitutes for expensive timbering, I leave to the mining operators themselves, as most competent to decide on the adaptations of each to special conditions. In this paper I shall confine myself to a brief discussion of the economies which may be exercised in the use of timber in mining, and the means in the power of the mine-operator for extending the supply of this material.

First, concerning the use of timber in the mine, it may be possible to be more careful in determining the necessary sizes, although the consideration of safety and the impracticability of calculating the strain exerted by the load which the timbers have to support, will hardly warrant us to look for much saving in that direction. A more probable gain of economy may be found in the selection of the kinds used for timbering, so that more durable kinds shall be employed, and the necessity for replacing reduced to a minimum. Even in this direction, there is but little hope, except, perhaps, in the more permanent constructions of shafts, winzes, etc. From the reports received by Mr. Dewey, it appears that the duration of different timbers, as well as of the same timber in different mines, is exceedingly variable, depending on the nature of the mine and especially its drainage-conditions and ventilation. But few reliable observations are obtainable under this head, by reason of the frequent changes of management and the general unconcern about the fate of timbering after stopes have been abandoned. The following list of species which, under all circumstances, may be expected to withstand decay most effectually, presents a series proceeding in general from the more durable to the less durable ones.

Eastern Range.

Conifers: Red Cedar, *Juniperus Virginiana*, L.; White Cedar, *Chamaecyparis sphoeroidea*, Spach.; Arbor Vitæ, *Thuya occidentalis*, L.; Bald Cypress, *Taxodium distichum*, Rich.; Long-leaved Pine, *Pinus palustris*, Miller; Red Pine, *Pinus resinosa*, Ait.; Cuban Pine, *Pinus cubensis*, Griseb.; Short-leaved Pine, *Pinus mitis*, Michx.

Broad-Leaved Trees: White Oak, *Quercus alba*, L.; Post Oak, *Quercus obtusiloba*, Michx.; Chestnut Oak, *Quercus prinus*, L.; Live Oak, *Quercus virens*, Ait.; Basket Oak, *Quercus Michaurii*, Nutt.; Burr Oak, *Quercus macrocarpa*, Michx.; Osage Orange, *Maclura aurantiaca*, Nutt.; Hardy Catalpa, *Catalpa speciosa*, Warder; Black Locust, *Robinia pseudacacia*, L.; Honey

Locust, *Gleditschia triacanthos*, L.; Red Mulberry, *Morus rubra*, L.; Chestnut, *Castanea vulgaris*, var. *Americana*, A. D. C.

Rocky Mountain Region.

Red Cedar, *Juniperus Virginiana*, L.; Pinyon Pine, *Pinus edulis*, Engelm.; Fox-tail Pine, *Pinus Balfouriana*, Murray; Douglas Spruce, *Pseudotsuga Douglasii*, Carr.; Western Larch, *Larix occidentalis*, Nutt.; Burr Oak, *Quercus macrocarpa*, Michx.

Pacific Slope.

Yew, *Taxus brevifolia*, Nutt.; Redwood, *Sequoia sempervirens*, Endlicher; Lawson's Cypress, *Chamaecyparis Lawsoniana*, Parl.; Canoe Cedar, *Thuya gigantea*, Nutt.; Douglas Spruce, *Pseudotsuga Douglasii*, Carr.; Western Larch, *Larix occidentalis*, Nutt.; Live Oak, *Quercus chrysolepis*, Liebm.; Post Oak, *Quercus Garryana*, Dougl.

In this connection it may be interesting to note also that of two pieces of wood of the same kind the heavier is the more durable; that quickly-grown hard woods, with wide annual rings, and slowly grown conifers with narrow (yet not too narrow) rings, and pines tapped for turpentine (on the tapped side), yield as a rule, other conditions being equal, the most durable wood. Coniferous wood from comparatively poor soils, high altitudes and dense forests, and hard woods from rich, deep, warm soils and isolated positions are, therefore, most durable. The resinous substances in conifers form an element of resistance to decay.

With regard to the influence which the time of felling exerts upon the durability of timber, it may be said that, with proper after-treatment of the wood, the time of felling seems not to affect its durability. Early winter-felling (December), however, should have the preference, because a smaller amount of fermentable sap is then in the trees. If the wood is cut "in the sap," it is more liable to fermentation. When it becomes necessary to cut in the leaf, it is advantageous to let the trees lie at full length until the leaves are thoroughly withered (two or three weeks) before cutting to size. With conifers this is good practice at any season; and, if it can be done, all winter-felled trees should be left lying to leaf out in spring, by which most of the sap is worked out and evaporated. If the timber is not to be used at once in the mine, and especially if it is to be used in dry mines, it is well to give attention to the treatment after felling and to seasoning.

Removing the bark from felled timber aids seasoning.

The logs should never be allowed to lie directly on the moist ground.

If winter-felled, the timber should be shaped to size within two weeks after felling and left placed on blocks—not upon the soil—in the wood; or if shaped at home, the sticks should be placed in a dry, *airy*—not windy—position, away from the sun and rain.

If dried too rapidly, wood warps and splits; the cracks collect water, and the timber is then easily attacked and destroyed by rot.

The application of processes of wood-preservation, for a short practical *résumé* of which I refer to Bulletin No. 1 of the Forestry Division, Department of Agriculture, may in certain localities and under certain conditions be warranted, at least for the more permanent structures, and in mines where conditions are specially favorable to decay. This will, of course, be a question of comparative cost, for the answer of which the table on page 107 of the publication mentioned will prove a welcome assistance.

With all economy in the use of material, however, the ultimate necessity of securing a reproduction of the supplies of timber by forest-management will not be obviated. This entails upon the mine-operators in the Western mountains, that they take an interest in the manner in which the timber on Government land is disposed of; in the regulations for its cutting; in the protection of the forest against fire and wasteful use; in fact, in a proper management of the public timber-domain.

The present lack of management, it is admitted by every honest observer, is most objectionable and unsatisfactory, wasteful in the highest degree, full of annoyance to the present consumers of the timber, and detrimental to all future interests. The fault does not lie so much with the executive power as with the legislative, which fails to recognize the need and the requirements of a proper forest-management, and, by inadequate appropriations for an insufficient protective service, wastes money and property, doing no good but, on the contrary, much harm, antagonizing present interests without benefiting future ones.

Before the present Congress there has been a bill appropriating $500,000 for the organization of a forest-management of the public timber lands, which has met the hearty endorsement of many prominent Western men, as particularly adapted to the conditions prevailing in the Rocky Mountain region, where the bulk of the public timber-domain is situated.

In spite of abundant and cumulative proof that no improvement in prevailing practices can be expected without thorough organization and proportionate expenditure, this bill has not only not received any consideration whatever in the House of Representatives, but a substitute, which does not remedy but increases the difficulties of the case, has been passed; and, without some exertion on the part of those most nearly interested in this legislation, namely, the Western mine-operators, they will suddenly find themselves worse off in regard to timber-supply than they have been hitherto, being forced to become depredators, or at least the abettors of thieves and depredators.

Where the mine-operator is owner of natural woodlands, as is not unfrequently the case in the East and South, he can do much to recuperate the ground cut over by simple, intelligent management. Such management consists in cutting with a view to favoring the natural reproduction of the desirable kinds. The present practice is to cut out the desirable timber wherever it is found, leaving inferior kinds and inferior trees standing; and these, by their shade and drip, prevent the growth of a new crop of better kinds. Removal of the inferior overtopping trees at the proper time, that is, after a young crop is secured, is often all that is necessary to remedy this evil.

It would be difficult, well-nigh impossible, in a short paper, to explain in detail the proper management of a wild forest for reproduction. This must needs differ in every locality, with every species, and offers, at times, complicated problems, the solution of which requires the knowledge and direction of an expert. In coniferous woods, which reproduce only from seed, it will be best to manage *en echelon*, *i.e.*, by clearing strips and leaving the trees standing to windward to seed the clearing. This may sometimes take several seasons, according to the frequency and proficiency of seed-years—for conifers, as a rule, do not seed every year plentifully. The breadth of the clearing—on which occasional reserves with not too spreading crowns may be left—should be made from 200 to 300 feet, according to the distance to which the wind can carry the seed easily.

In deciduous woods, which reproduce themselves from the stump as well as from seed, care must be taken to keep in check the inferior kinds, which are usually more rapid growers, seed more plentifully, and, if given full enjoyment of light, soon overtop and impede in growth, or crowd out entirely, their slower neighbors.

In managing such deciduous growths, it is necessary to cut out the inferior timber a few years before the main crop is utilized, thus giving preponderance and better chance for seed-production to the remaining growth, and, as soon as the young growth has established itself, to cut away the old. This may entail a cutting of more timber than is required for the moment; but the seasoning of the timber before use will enhance its value.

How by judicious thinning the young growth may be accelerated, and wild brush-lands may be improved, I cannot stop to discuss.

Lastly, we have the case of a mine-operator owning tracts of barren land from which the timber has been removed, leaving fire and cattle to extinguish every trace of forest life and turn them into useless wastes. Such sights are most familiar around the coal-mines of Pennsylvania. These areas should and could be profitably utilized by the owner in forest-planting.

The first question is, What to plant?

Not only must it be a durable timber, capable of growing well on poor land, but one which will attain desirable quality and size in a comparatively short time, and will be easy of management and reproduction.

Various localities and climates will vary the selection. Taking the coal-regions of Pennsylvania for an example, we may say that, of the timbers mentioned in the list above, the black locust, chestnut and catalpa, with an occasional oak and cedar, will answer.

The black locust is a very rapid grower, and develops into sufficient size for prop timber in a very short time. Twenty to thirty years will yield, on a tolerably suitable location, lengths of 20 to 30 feet, 4 inches in diameter at the small end. For strength (resistance to longitudinal compression especially) it stands first among the commoner Eastern timbers (crushing-weight, 694; modulus of rupture, 1273; those of white-oak being 511 and 905 respectively). It is very easily handled in planting by cuttings, suckers, stakes, seedlings, or seed, and reproduces itself readily by sprouting from the stock. There are two objections to its use in forest-planting by itself: the one, its liability to attacks by borers; the other, its thin foliage, which fails to enrich the soil and to shade it against evaporation—two desirable requisites on poor soils.

The first objection is largely mitigated, and the last is overcome, by mixing the locust with other shady kinds. No better associate could be found than chestnut or catalpa. The former, although inferior in regard to strength (modulus of rupture, 696; resistance

to longitudinal pressure, 381), is most durable and useful for ties and in other shapes, producing rapidly large quantities of wood per acre, and, by its shade and foliage, benefiting the soil. The catalpa, which, as regards strength, stands near the chestnut, recommends itself by the ease of its propagation, adaptation to unfavorable conditions, and rapidity of growth.

I would propose to start a plantation by planting rows of black locust, alternating with either chestnut or catalpa, 4 feet apart, occasionally adding one of the white oaks, say at the rate of 100 per acre. The latter would not be of much value when the other crop was fit to be utilized, but will enhance the value of the plantation for future years without adding to the cost of production.

There will be needed some 1300 locust seedlings, 1000 catalpas or other shady timber species, and one bushel of chestnuts to the acre. The plant-material, if bought from a nurseryman, may cost $25, but can be procured at probably half that cost.

In fact, there are at present nurserymen in the West who contract to plant and insure a stand of forest growth at the rate of $25 to $30 per acre. It can be done on a large scale for less, especially in the East, where climatic conditions are favorable to tree growth and less work is required to insure success.

As to the profitableness of such a plantation, there cannot be the shadow of a doubt. Including expenses for superintendence, and compound interest at the rate of six per cent. on the original outlay (the land, being waste land, may be left out of the calculation), one acre of growth may be said to have cost, at the end of twenty years, about $100. Disregarding all the material which may have been obtained by thinnings during that period, and leaving out the value represented by the chestnut and other accessory growth on the ground, there can safely be cut at that time not less than 20,000 linear feet of locust post-timber over four inches in diameter (more *has* been cut), making the cost per foot not more than one-half cent, while such timber cannot now be bought for less than two to three cents per linear foot (in New York City eight cents per foot is paid at wholesale). Meanwhile there will have been inferior material realized by thinnings; the chestnut will have grown into useful size, and may be cut into rails, ties, posts, etc.; and, besides, other valuable growths will be on the ground.

No artificial renewal of the locust growth is necessary. From each stump three or four vigorous sprouts will grow and will be ready for the same use in less time than the original seedlings, since

the sprouts grow more rapidly. All the care that is necessary is to keep out fire and cattle for the first three or four years; and occasional thinning, to give air and light to the timber which is to be favored.

A hundred-acre plantation of this description may be compared in value to an investment yielding an intermittent annuity, every twenty years, of not less than $40,000; or, capitalized, it represents an investment of $50,000, yielding such annuity at three per cent. interest, as safe as United States Government bonds, while it has cost less than $5000; that is to say, one need not raise more than ten per cent. of the stock-value to start the enterprise.

A systematic proceeding for a continuous concern, like some of our collieries, would be to plant every year, or periodically, as many acres as are needed to produce the yearly or periodical supply of posts. Before this has been continued for a series of twenty years, a never-failing supply of post-timber will be secured, since the thinnings of the first plantations will begin to furnish the needed material at an earlier date.

It is impossible to detail further the particulars of such an enterprise. Local conditions must needs vary these. My object in this paper has been simply to direct the attention of mine-owners and operators to the possibilities and opportunities which lie open to them with regard to their timber-supply.

NOTES ON THE ROASTING OF THE HUDSON RIVER CARBONATES.

BY INGERSOLL OLMSTED, BURDEN, N. Y.

(Buffalo Meeting, October, 1888.)

THESE ores are of two classes, Bessemer and non-Bessemer, existing in separate, though adjoining, beds. Both are carbonates, with small admixtures of oxides and other combinations.

To prepare them for market they are roasted in kilns, which have already been described by Mr. John Birkinbine at the Troy Meeting, October, 1883 (*Trans.*, xii., 378). Concerning their size, arrangement, etc., I need, therefore, only repeat that they are 60 feet in height; have wrought-iron shells 24 feet in diameter; are lined with fire-brick, and supported by cast-iron mantel and columns.

As originally built, the draught was wholly from the bottom; but afterwards two rows of small circular holes were cut in them, a little short of half way up, and encircling the kiln. The kilns are ten in number, with an estimated roasting-capacity of about 100 tons daily each.

The richer of the two classes of ore is a true Bessemer, containing only about 0.035 per cent. of phosphorus. It is uniform in character, and unmarked by any specially interesting features. In the present paper I purpose to speak mainly of the non-Bessemer ores. From my own notes I am not able to furnish many complete analyses, such being seldom required of me; but I subjoin two, which may serve to show the character of the two classes.

Analyses of Roasted Ores.

	Bessemer. I.		Non-Bessemer. II.
Silica,	9.20		17.18
Carbonic acid and water,	.55		undet.
Iron oxide, FeO,	2.85	Bisulphide iron	1.47
Iron oxide, Fe_2O_3	63.03		65.73
Alumina,	2.19		3.02
Lime,	6.80		4.35
Magnesia,	7.13		5.72
Oxide of manganese,	3.90		2.66
Phosphoric acid,	.102		.319
Sulphur,	1.788		.....
Oxygen with sulphur,	1.58		
	99.12		100.449
Metallic iron,	46.34		46.70
Phosphorus,	.044		.139
Sulphur,	.....		.786

The foregoing analyses do not adequately indicate the complexity of the ore, which, indeed, presents something of a puzzle to the analyst, containing, as I believe all samples of the non-Bessemer class do, all the following elements in all the combinations named, viz.: *Phosphorus* (apatite, phosphoric acid). *Sulphur* (sulphuric acid, iron bisulphide, calcium sulphate). *Iron* (metallic iron, magnetic oxide, sesquioxide, bisulphide). *Calcium* (oxide, phosphate, sulphate, hydrate). *Lead* (native), and *Silica* (native, silicate of alumina); besides, probably, traces at times of titanium, arsenic, and rarer metals. The ore always contains a considerable amount of water—sometimes, when it has been exposed to rain, as much as 8 per cent.

It is but fair to say that the sulphur in I. is much higher than the average, and that the iron in II. is a trifle higher than I have generally found.

The ore as drawn from the kilns presents a varied appearance both in color and form. The color is mainly chocolate, but lightening into many shades of red and gray, or deepening into a blue-black. These appear to arise from variations in heat and atmospheric effects in the kilns, not from varying percentages of iron, etc.; and they furnish but little guide to the eye in judging of the nature and richness of the ore.

For instance, a large lump of irregular form, having a light-red exterior and a dark greenish center, was brought to me some time ago with the request that I would, if possible, discover why the outside only was roasted. A test, however, proved at once that it was thoroughly roasted throughout, the dark center having misled the sorter. The percentage of iron was low, 27.45 per cent. in the exterior part, and practically the same in the center. Another lump, almost precisely similar in appearance, gave 56 per cent. of iron throughout.

As far as my observation goes, the ores, with two exceptions, are magnetic after roasting; that is, when powdered they will separate from the silica and earthy matters mixed with them and adhere to an ordinary bar-magnet. The first exception to this is the dark center of the lump above mentioned, which was entirely non-magnetic until powdered and heated in a platinum crucible, when it became light-red in color and magnetic. The exterior of the lump was magnetic. When I exposed a piece of the dark center to the flame of a gasolene lamp the red magnetic skin appeared, but the center remained dark, thus repeating the history of the main lump. The second exception I shall mention later. The lowest heat, still sufficient to roast thoroughly which I have been able to give, has always rendered the sample magnetic throughout; but whether the exact degree necessary for the expulsion of the carbonic oxide will do so or not I am unable to state. This magnetic property has suggested to my mind the possibility of producing a separated ore by an electric separator, but there are many difficulties in the way, the chief, I think, being the fine grain of the ore and the intimate mixture of silica and impurities, which would necessitate very fine crushing.

The form of the roasted ore is varied, as I have said; but examination shows four main characteristics, which may be classified thus:

I. A rough, shapeless mass;
II. A fine-grained, flat slate;
III. A mixture of white silica-grains with red or black ore-grains, occurring mainly in thick flat slabs;
IV. "Clinkers," so-called.

The analysis of many pieces of each class points to the rough generalization that Nos. I. and IV. are the richest; the others mostly lean, though stray pieces of one class occasionally approach those of another so closely in percentage of iron that the line becomes hard to draw.

Class IV. presents the most interesting features. For some reason not yet clear these clinkers form in the kilns and, descending to the gates, give trouble by being too large to pass through, and requiring blasting. They are, in the main, roughly rounded masses, from 18 inches to 3 feet in diameter, composed of pieces of ore, rock, coal-ashes, etc., cemented together with fused ore, or something similar. But there are many of a different nature. One appears as though formed from a single ore-lump; it is nearly gray in color; very hard; has a glazed surface, from which the hammer rebounds sharply; and occurs in lumps the size of a man's head, attached to other clinkers, but plainly of separate nature. In the mortar it yields a yellowish powder, separable by the magnet into a yellow-gray earth and a sharp, black, glittering sand. The earth contains about 42 per cent. of metallic iron, and is the second exception to the general rule I have stated, that all the roasted ore is magnetic. The black sand contains about 53 per cent. of metallic iron.

Another clinker is apparently also from a single ore-lump, but has no evidences of fusion, no glazed exterior, and retains its original form. It is very hard and heavy, presents a glistening-black fracture with evidence of sulphur, and often contains 60 per cent. of metallic iron.

Another resembles a slag-finger seen at the tuyere of a chilling-furnace, or an ore-stalactite formed on the bottom of something and then broken off. These I have occasionally seen forming at the kiln-gates, the fused matter trickling down the wall when the kiln was "in trouble," a condition corresponding to a scaffolding furnace.

A prominent characteristic of the "cemented" clinkers is that they often contain pieces of Class III.; but such pieces show no evidences of fusion or alteration, and are only stuck to the rest of the mass. In spite of close study with a magnifying glass I have not

been able to connect any other particular form of ore, the slate, for example, with this clinker-formation, nor have I ever seen any fused or altered rock. Moreover, the lumps of ore never seem to show the beginning of fusion arrested there. All these facts point to complete change taking place in one kind of ore.

The reason for this clinker-formation remains an open question, as I have said. Too much heat has been the cause commonly assigned; but that of itself would not produce the metallic iron visible in them. To test the action of heat upon the ore I placed some, in inch-cube size mixed with coal, in a Hessian crucible, and exposed it to the full heat of a gasoline furnace, keeping the crucible at an almost white heat for hours, but failed to produce any effect except thorough roasting—this, too, although I varied the conditions, with crucible open and closed, by charging wet ore and coal-dust, etc.

I then considered that there must be some action of oxidizing and reducing gases at the bottom of the trouble; that it might be promoted by the ore remaining longer in or near the fires than is necessary to roast it completely. This might easily happen, since the drawing of the kilns is regulated by the appearance of hot ore at the gates, the convenience of shipment and other causes, not by the time necessary to roast. I do not know that that length of time has ever been precisely determined here. Clinkering might also arise, I thought, from some of the ore fusing or reducing more easily than the rest, and thus running in the fire necessary to roast the more refractory.

In our non-Bessemer vein there occur constantly seams of black slate, generally narrow. It is very siliceous, fine-grained, nearly black between the gray natural ore, breaks into thin plates, and furnishes my Class II. Beside its carbonate of iron it carries from 1 to 15 per cent. of some other iron combination, insoluble in concentrated boiling acid. May not this fuse readily and, under some conditions, cause trouble? To follow up the investigation on this line, I designed a special apparatus for roasting samples in the presence of different mixtures of oxidizing and reducing gases, but have not been able as yet to carry on the experiment.

It is on this point that I invite discussion, namely, the reason for the formation of clinkers described. Perhaps I ought to add that our fuel is good culm, and the charge fixed by practice as best, is about 1 ton of fuel to 30 of ore.

At certain times these clinkers form in large numbers and virtually scaffold the kiln, which decreases its output, becomes hot through-

out, and is said to be "in trouble." Then they are termed "bad" clinkers, and are thrown out on account of the sulphur they contain. The average of this objectionable element in the ore, including clinkers, is from 0.70 to 0.90 per cent.; but the clinkers alone usually show over 2 per cent. Now the question which I wish specially to recommend to the attention of furnace-men is, What effect will charging these masses among our friable and easily reducible ores have upon the furnace and its work? Would they need more exposure to the reducing gases than the porous ore, and thus, by descending low in the furnace while yet partially unreduced, cause irregularity of working? Would it be advisable to reject them, supposing them to constitute a small portion of the ore only?

The sulphur in our roasted ore has generally been considered to exist as sulphuric oxide. I think Mr. Sherrard, of Scranton, holds this view. It is also said to roast off in the furnace and not to affect the iron. Now, would this hold true of the clinkers? Is not the sulphur more likely to be in them as pyrites, and, in such hard masses, to descend into the zone of fusion? I regret that it is impossible to give any complete analyses of these clinkers which might aid much in the solution of such questions. I have only a few fragmentary tests for iron and silica, which do not show anything.

Apparently, the ore was designed to confound many pet blast-furnace theories. For instance, I was once told that it would not make a large percentage of No. 1 or No. 2 iron, because of its manganese (2 to 4 per cent. of the black oxide) and sulphur; but, since then I have seen a furnace, running on this non-Bessemer ore alone, make nothing but No. 1 iron, for days together. In that iron I found from 1.75 to 2.25 per cent. metallic manganese, and only a trace of sulphur. The iron, when raw, was covered with the cloud of sparks, which ordinarily accompanies cold iron, and the slag had a chocolate-colored or nearly black surface, yet contained, nevertheless, only a trace of iron. The fumes at casting were remarkably dense and sulphurous, and I, more than once, noticed in them the odor of cyanides.

The iron made exceptionally good castings, and pleased the foundry-men very much.

The variety in appearance and in composition of the samples of this non-Bessemer ore is very striking. Upon my table lie upwards of a hundred lumps, all tested and tagged, with the percentage of iron at least, often with that of silica and manganese. The silica in many is plainly visible to the eye, in pure white grains, scattered

through the mass, or in narrow veins which, under the action of fire, become brittle and split the mass throughout, leaving the white crystals upon one face. In others, as in the black slate, the silica is entirely in combination. The percentage of iron varies from 7 to 56, though, even to the trained eye, there is very little difference in appearance. Some contain so much lime, that, on mere exposure to the atmosphere, they have slacked into a heap of dust, though they carried over 40 per cent. metallic iron.

These peculiarities have occasioned much trouble with our customers, and complaints of rock, dust, careless preparation, etc., from those who are ignorant of the originally mixed condition of the ore in the vein, and the confusion unavoidably resulting from the roasting and consequent alteration of form. A knowledge of them, together with a laboratory accident, led me to make the following experiment:

A large sample of non-Bessemer ore was reduced to a coarse powder, divided into halves, and these marked, respectively, Nos. I. and II.

No. I., without any further preparation, was passed over a 60-inch sieve, and the resulting powder marked A, the fine ore, tailings, etc., being marked B. No. II. and A and B were then carefully sampled and analyzed with the following results:

	II.	A.	B.
Iron,	39.06	44.51	37.70
Silica,	23.95	18.40	27.
Phosphorus,	0.201	0.120	0.270

A partial repetition of the experiment, the analysis of II. being omitted, gave

	"A."	"B."
Iron,	43.05	36.92
Silica,	20.53	29.53
Phosphorus,	0.102	0 183

The samples originally taken did not fairly represent the ore, being leaner than the average; but this did not defeat my intention, which was to learn whether a richer separated ore could be obtained by such screening.

If, as my experience goes to show, the soft ore is the rich and desirable, then the difference would have been more marked, had the proper proportion of rich ore existed in the samples. The experiment has been well confirmed, and seems to show that our leaner,

more siliceous, non-Bessemer ore really consists of a mixture of rich ore with veins and lumps of lean, phosphatic stuff. There is no such difference visible in the natural ore, nor do the tests of diamond drill cores and daily drill-samples from the mines encourage the belief.

I am, nevertheless, inclined to cling to the theory, in part at least, and to consider the other disagreeing results to arise from the intimacy of the mixture, rendering a drill-core witness only as to the ore it actually contains.

THE EFFECT OF VELOCITY AND TENSION OF GASES ON THE REDUCTION OF ORES IN THE BLAST-FURNACE.

BY THEO. W. ROBINSON, JOLIET, ILL.

(Buffalo Meeting, October, 1888.)

THE evolution of the modern blast-furnace from the embryonic stages of comparatively few years ago, has been the work of wide practice and experiment. That much is still to be desired, the experience of every furnace-manager will testify.

Mr. Walsh's paper on "The Irregularities of the Blast-Furnace Process, and a Practical Way to Avoid Them" (*Trans.*, xv., 419), proposes a new solution of the problem of interior lines, and one that has given rise to not a little speculation and controversy. The investigation described in this paper was undertaken to test experimentally the theory of Mr. Walsh as to maximum reducing capacity.

Briefly explained, Mr. Walsh's views (outside of his claim for increased efficiency due to having the bosh wholly within the melting-zone) are, that a furnace of contracted stack and throat, on his lines, may equal in product one of considerably greater cubical capacity. The explanation offered for the theoretical greater capacity per cubic foot of his furnace is, that the increased tension of gases caused by passing through the small area will promote transpiration, thus facilitating reduction and permitting more rapid driving.

This alleged relation will appear from a hypothetical case in which a furnace, A, constructed on Mr. Walsh's lines, has but $\frac{1}{2}$ of the cubical contents of a furnace, B, of modern type; the two being equal in height. In order that A shall have the same capacity as B, both working normally with the same mixture and burden, and

amount of blast per unit of time, we must have the following relative conditions presented.

1. Since A holds only $\frac{1}{x}$ as much material as B, the ore must be reduced x times as fast, and hence can remain exposed to the action of the gas only $\frac{1}{x}$ as long, in A as in B. In short, A must be driven with x times the rapidity of B.

2. With the same amount of blast per unit of time in both furnaces it follows that in A, with but $\frac{1}{x}$ the area of B, the velocity produced must be x times that in B, and consequently (neglecting the slow downward movement of the charge, as relatively nothing to the rapid ascent of the gas) each particle of ore in A must be impinged upon by x times as many atoms of gas in a unit of time as is the case in B.

With the given conditions of equal blast, burden and product, the above is a correct statement of the relation existing between a Walsh furnace and a furnace of usual lines. For illustration, take the comparison cited by him of his furnace of 10,000 cubic feet capacity, and the North Chicago of 14,694 cubic feet. On the basis of equal product, the former must be driven nearly one and one-half times as fast as the North Chicago, the velocity of the gases will be nearly $1\frac{1}{2}$ times as great, and the tension over twice as great. With the latter working at its actual maximum efficiency, this would require exceedingly rapid driving; and Mr. Walsh relies for a successful issue of what would be phenomenally quick reduction on the results of Sir Lowthian Bell's experiments pertaining to the relative reducing effect on ores of gases of different velocities. These showed that the faster the current, the greater the reduction. They were performed by passing different amounts of gas in a unit of time over various samples of ore at a pretty uniform temperature of 770° F. Mr. Bell's interpretation of the results is.

"On the whole I am inclined to the belief that some slight elevation of temperature accompanies the rapid passage of the gas, owing to a more energetic action than takes place with the slower current, and that this rise of temperature in its turn has promoted chemical action."

On the other hand, as before stated, Mr. Walsh advances the theory that the increased effect is caused by the more rapid transpiration of gases, produced by the increased pressure of the rapid current. In the experiments of Mr. Bell, just quoted, increased reduction by the rapid current may be influenced by one or all of the following factors:

1. The presence of a different amount of reducing gas.

2. The more thorough removal of the retarding CO_2.

3. A higher temperature, arising from the rapid current enhancing the chemical action.

4. The increased tension produced by the greater velocity causing a more rapid penetration of the gases into the ore, according to the law of transpiration.

In order to interpret intelligently in their application to practice, the results of these experiments, it is necessary to ascertain first, wherein the conditions within the furnace and in the experiments correspond, and to what extent they differ.

Remembering that each furnace has the same amount of blast per unit of time, we see at once that factor No. 1 can have no bearing on the matter. Though the velocity of the gas in A is, say, twice as great as in B, a given weight of ore is subjected to the action of the same amount of gas in a unit of time, in each case. Moreover, since, by the premises, B is working with the greatest economy of fuel, it is plain that the gases in A under an equal efficiency must have the same ratio of CO_2 to CO as in B. This disposes of factor No. 2, and leaves Nos. 3 and 4 as the only conditions in Bell's experiments compatible with the situation of the ore in A, and as possibly explanatory of a more rapid reduction. It is important, therefore, to examine experimentally the effects on the reduction of iron-ore of two conditions, namely, the velocity and the pressure of the reducing gases.

It is impracticable to include in such an experiment the whole series of chemical changes exhibited in a blast-furnace, unless the furnace itself be employed as the apparatus for the experiment. Neither Mr. Bell's experiments on this particular point, nor those described in this paper, were thus conducted; and it must be left to the judgment of competent critics, whether the conclusions to which the latter point are perfectly applicable to the larger operations of practice.

Being unable to compare experimentally the two whole furnaces, A and B, I have thought that attention might be fixed upon two equal masses of ore, in corresponding parts of the two furnaces, within the zone of reduction. The same relation as to rate of reduction, temperature, velocity and tension of gases, should here exist as throughout the entire furnaces, the only dissimilarity (so far as I perceive) lying in the fact that by virtue of its constricted area the layer *a*, in A, has twice the thickness of layer *b*, in B. Assuming

a thickness so small that the variation in temperature for the top and bottom will be inappreciable, we have presented a practical example that may be copied by experiment. Upon this assumption the experiments now to be described were conducted.

In order to follow as closely as possible the actual conditions presented in the furnaces, furnace gas-was employed as the reducing medium. The experiments were performed in duplicate apparatus, one set representative of the situation in A, the other in B. Two similar gasometers of something over 100 liters capacity were constructed, each with a water-gauge on the side, which upon careful calibration permitted the accurate measurement of the gas used in each operation. The supply of gas was obtained by tapping Ethel furnace No. 1 of the Joliet Steel Co., about 17 feet above the tuyeres, and by means of an iron pipe, fitted with a perforated cork, holding a large Y-tube, and connected with the gasometers by rubber tubing, a gas of uniform composition was obtained for each. I would note that the best results were obtained by allowing the gas to displace the air in the gasometer by its own pressure, no water being employed.

The temperature decided upon was that of melting zinc, about 417° C. or 782° F., obtained and applied by means of a bath of melted lead, held in a cast-iron pot, heated by gas. This was of such a size as to allow a distance of two or three inches to intervene between the sides and the ore-holding apparatus; and a grating, fastened to the bottom, prevented contact beneath. Thus extreme variations in temperature, by proximity to the sources of heat were prevented. The temperature of the bath, as tested by immersing pieces of zinc and antimony in an iron tube, was pretty uniformly that of softening zinc.

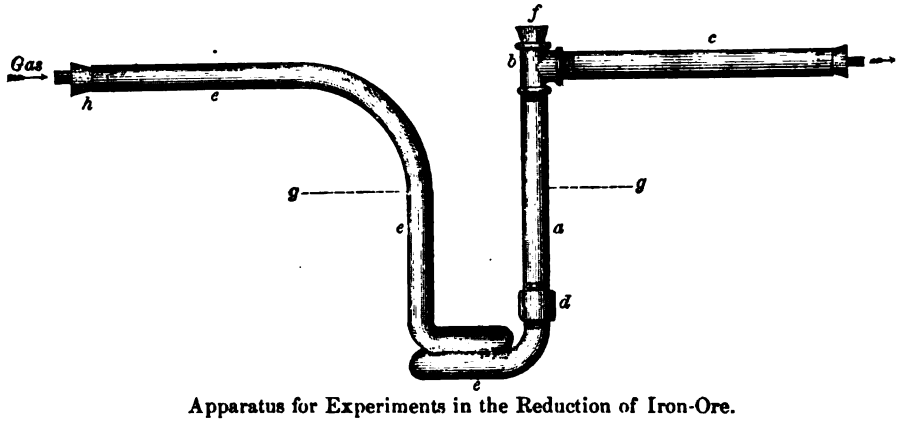
The best mode of exposing the ore to the gas was a matter of some deliberation and trial. With the exception of one line of experiments, all reductions were performed by means of the apparatus shown in Fig. 1, in which *a* is a piece of iron pipe, about 8 inches long with the cross of a T screwed on one end at *b*, from the arm of which projects another section of pipe, *c*, while at the other end *d*, it is connected with a pipe, *e*, bent in a horizontal spiral, like a worm, and then rising vertically, parallel with the first. Between *a* and *e* are inserted a couple of thicknesses of fine platinum gauze, which the coupling holds firmly in place and with an air-tight joint. The upper part of the T is closed by means of a cork, *f*, the removal

of which allows the introduction of the ore, which then rests on the gauze at *d*.

During the experiments, the apparatus was immersed in melted lead, up to the line *g*, *g*.

The gas was applied at *h*, and then passed down, twice around and up before coming in contact with the ore—a distance in all of about

FIG. 1.



Apparatus for Experiments in the Reduction of Iron-Ore.

30 inches—to prevent a cooling action on the ore by permitting the gas to acquire the temperature of the bath. In all cases the two sets of apparatus in use were placed with one coil within the other, the vertical parts holding the ore being brought side by side, to prevent different temperatures by variations in different parts of the bath. When inserted in the bath, they were kept in place by weights.

In the one line of experiments referred to, where the form of apparatus just described was objectionable, two similar round cast-iron boxes, 3 inches in diameter, fitted with entrance and exit-tubes. were used, being placed one directly over the other, in the same bath.

The ore employed was a very uniform sample of soft Gogebic hematite, crushed to pieces about the size of half a grain of wheat, by passing through a 12-mesh, and catching on a 15-mesh sieve, With the object of determining by titration the amount of reduction, the ore in the first experiments was thoroughly heated in a glass-tube, to remove water and other volatile components, and, though titration failed of completeness, by reason of the coarseness of the samples, the ore was in all cases subjected to this treatment, as a method of comparison with the work gone over before.

In determining the amount of reduction, various methods were tried. The following were only partially successful; oxidation of the reduced ore by oxygen in an open platinum dish; oxidation in a Gooch crucible, fastened on top of an iron pipe, by titration of the raw and reduced ore; and analysis of the waste gases. In the first two cases, perfect oxidation proved well-nigh impossible, and titration failed through inability to obtain similarity of samples. Examination of the waste gases as a basis of calculation was abandoned on account of the trouble experienced in getting a good average sample. The method finally followed was, after thoroughly mixing the whole of the reduced sample by grinding fine in an agate mortar, to oxidize a known weight of it by oxygen in a platinum boat, inserted in a glass tube with a layer of copper oxide behind, catch the CO_2 in caustic potash, and weigh the ore, now completely oxidized to Fe_2O_3. Care was taken at all times to regulate the heat so as to cause no incipient fusion of the oxides.

The calculation is easy. From the CO_2 produced, the carbon in the weight taken of $Fe_xO_y + C$ (reduced material) is known, and by difference, the amount of Fe_xO_y. Subtracting the weight of Fe_xO_y from that of the Fe_2O_3 produced, we have the amount of oxygen lost in reduction.

The first series of experiments was made to compare the effect of equal amounts of gas, with the same rate of flow and tension, upon equal masses of ore, subjected to a like temperature, under conditions comparable with the layers of ore in the furnaces under discussion.

Two pieces, one of $\frac{1}{2}$-inch and the other of $\frac{3}{4}$-inch iron-pipe, having internal areas nearly as one to two, were fitted with platinum gauze and arranged as shown in Fig. 1. These, loaded with their charge of ore, were immersed in the bath, and, after allowing a sufficient length of time for the attainment of its temperature were connected with their respective gasometers. To counteract the effect of any error from calibration or otherwise, each gasometer was connected alternately with the large and small tube. The small tube, having half the capacity of the other, contained a column of ore of double the height and about half the sectional area of the larger. Now, by passing the same volume of gas in a given time through each, the relation of one to the other is the same as exists in the two layers, in furnaces A and B, with the exception that the tension of the gases is practically that of the atmosphere in both. Trial of the loaded tubes by a manometer, at a rate of from 30 to

40 liters per hour, showed a pressure corresponding to less than 2 millimeters of mercury. Hence, in the following table of experi-

TABLE I.

No. of Experim't.	Apparatus.	Amount of Ore in Grammes.	Amount of Gas in Liters.	Per cent. by Volume of CO in Gas.	Duration of Experiment in Hours.	Rate per Liter per Hour.	Per cent. of Reduction.	Per cent. of Deposited Carbon.	Ratio of Deposited Carbon to Reduction.
1	L	20	40	31.20	4.75	8.42	9.47	4.13	43.61
	S	20	38		4.75	8.00	9.67	2.36	24.41
2	L	20	43	31.93	3.50	12.28	11.57	3.94	34.05
	S	20	43		3.50	12.28	11.44	2.73	23.86
3	L	20	42	31.80	1.5	28.00	13 06	4.54	34.76
	S	20	42		1.5	28.00	11.09	4.38	39.49
4	L	20	68	31.60	5	13.60	14.66	5.49	37.44
	S	20	68		5	13.60	13.93	4.43	31.80
5	L	20	80	30.40	6	13.33	15.33	10.39	67.78
	S	20	80		6	13.31	10.00	1.90	19.00
6	L	10	67	30.00	7	9.57	27.57	8.21	29.77
	S	10	67		7	9.57	26.91	8.56	31.80
7	L	10	66	.....	3.5	18.85	14.93	1.30	8.70
	S	10	66		3.5	18.85	9.85	.35	3.55
Aver.	L	17.14	58	31.15	4.46	14.86	15.22	5.43	36.59
Aver.	S	17.14	57.71		4.46	14.80	13.27	3.53	24.84

ments, the amount of ore and gas, tension, rate of flow, and temperature are constant, with velocity due to the constricted area of the small tube *a* variable.

The results obtained are recorded in Table I. The amount of reduction in all experiments is calculated on 100 parts of oxygen, and the deposited carbon on 100 parts of the reduced substance. Under the head of apparatus, L and S refer to large and small tubes respectively.

In interpreting this and the following tables, only marked and average differences should be noted. Greater or less variation in temperature is certain to exist; a deposition of carbon, occurring to a greater extent in some experiments than others, will prevent as perfect contact of the gas with the particles of the ore in one case as in another; and the effect of such slight changes is sometimes very marked.

The general result derived from these experiments is that under conditions of equal ore, gas, tension, temperature and rate of flow, with velocity twice as great in the small as in the large tube, the reduction obtained was nearly 2 per cent. higher with the slow than with the rapid current. If increased velocity *per se* could not accelerate, I can see no theoretical reason why it should retard reducing action; and I am constrained to believe that such differences as greater height of the ore-column and hence closer proximity to the upper and cooler portion of the bath and possibly less perfect absorption of heat by the more rapid current, might be the cause of the lower results in the small tube. To test the latter point, thermometers, with their bulbs protected from contact with the sides of the pipes by a platinum spiral, were introduced just over the ore in each tube. No constant or appreciable differences were discovered. The fact is thus demonstrated that, under the conditions employed, increased velocity, if at all enhancing reduction, does not produce sufficient action to prevent its being masked by the effect of slight and unavoidable discrepancies in the mode of determination. For these reasons, it is unsafe to attempt to draw conclusions from individual and slight variations in the results, unless verified by more extended research.

To investigate the effect of velocity under conditions different from those represented in Table I., the experiments embodied in Table II. were undertaken. The apparatus used was the same as that employed in the experiment just discussed, except that No. 2 tube was of the same diameter and capacity as No. 1 (corresponding to the large tube of Table I.). Here the amount of ore and gas, tension and temperature are constants, with velocity due to increased flow per unit of time a variable. The tension was that of the atmosphere. The tubes being

identical, the height of the column of ore is the same, and thus eliminates one of the sources of error in the former case. Another dissimilarity is that in this case each particle of ore is impinged upon by the same number of atoms of gas in each tube, though in No. 1 with twice the velocity of No. 2. It will be remembered that where the small tube was used each particle of ore was brought in contact with twice as many atoms of gas as in the large tube, and with twice the velocity.

TABLE II.

No. of Experim't.	Apparatus.	Amount of Ore in Grammes.	Amount of Gas in Liters.	Per cent. by Volume of CO in Gas.	Time of Experiment in Hours.	Rate per Liter per Hour.	Per cent. of Reduction.	Per cent. of Deposited Carbon.	Ratio of Deposited Carbon to Reduction.
8	No. 1.	20	62	32.10	2.50	24.8	15.73	1.70	10.80
	No. 2.	20	62		5.00	12.40	22.30	5.68	25.47
9	No. 1.	10	65	31.20	3.75	17.33	24.17	14.04	58.08
	No. 2.	10	65		5 50	11.82	21.72	16.30	75.50
10	No. 1.	10	17	.....	.75	22.66	13.12	.88	6.70
	No. 2.	10	17		1.50	11.33	16.32	1.71	10.48
Aver.	No. 1.	13.33	48	31.65	2.33	21.59	17.67	5.54	25.19
Aver.	No. 2.	13.33	48		4.00	11.85	20.11	7.89	37.15

These results approximate those of the first table as regards effects of velocity on reduction, and tend to confirm them. In Table I., the slow current as an average gave nearly 2 per cent. greater reduction than the fast, and in Table II., the difference is about 2½ per cent. The anomaly presented by experiment No. 9 may very likely be accounted for by the fact that tube No. 2 got plugged up twice by carbon, and had to be cleared with a reamer.

The second series of experiments was undertaken to ascertain the effect of pressure on the action of gas upon ore. The apparatus adopted was two round cast-iron boxes, 3 inches by 2½ inches inside diameter, fitted with entrance- and exit-tubes, and having covers

fastened in place by key-bolts, making, with asbestos packing, an air-tight joint. After charging with ore, they were wired one on top of the other, and immersed in the bath. Tension was produced by passing the gas through a mercury column and a sufficient head of water, obtained by connecting with a movable elevated reservoir. The apparatus otherwise was the same as in former experiments. The conditions maintained were ore, gas, velocity and temperature as constants, with tension of gas *a* variable.

Table III contains the results.

TABLE III.

No. of Experim't.	Apparatus.	Amount of Ore in Grammes.	Amount of Gas in Liters.	Per cent. by Volume of CO in Gas.	Pressure per Square Inch.	Time of Experiment in Hours.	Rate per Liter per Hour.	Per cent. of Reduction.	Per cent. of Deposited Carbon.	Ratio of Deposited Carbon to Reduction.
11	Box 1.	10	58	27.00	2 lbs.	4.50	12.89	11.73	9.23	78.68
	Box 2.	10	58		Atmospheric.	4.50	12.89	7.88	6.87	87.18
12	Box 1.	10	58	31.20	2 lbs.	4	14.50	11.55	3.22	27.87
	Box 2.	10	58		Atmospheric.	4	14.50	7.80	2.12	27.18
13	Box 1.	20	70	.....	Atmospheric.	5.50	12.72	13.24	5.76	43.50
	Box 2.	20	70		2 lbs.	5.50	12.72	10.37	7.38	71.16
14	Box 1.	10	25	35.60	Atmospheric.	2.25	11.11	10.25	1.85	18.04
	Box 2.	10	25		2 lbs.	2.25	11.11	11.19	3.26	29.13
Aver.		12.5	52.75	31.26	2 lbs.	4.06	12.80	11.21	5.77	51.71
Aver.		12.5	52.75		Atmospheric.	4 06	12.80	9.79	4.15	43.97

These results show pretty conclusively that increase in pressure, *per se*, other things equal, causes the gas to have a greater reducing action. To equalize the effect of any differences that might exist in temperature between the top and the bottom box, the pressure was in 11 and 12 applied to No. 1 and in 13 and 14 applied to No.

2. There is some question whether the results recorded under experiment 13 did not get changed about in analysis; and the testimony of the other results would so indicate. To further test the influence of tension, the two experiments given in Table IV. were made. The apparatus used and the manner of application was the same as in the first series of experiments given in Table I.

TABLE IV.

No. of Experim't.	Apparatus.	Amount of Ore per Grammes.	Amount of Gas in Liters.	Per cent. by Volume of CO in Gas.	Pressure per Square Inch.	Time of Experiment in Hours.	Rate per Liter per Hour.	Per cent. of Reduction.	Per cent. of Deposited Carbon.	Ratio of Deposited Carbon to Reduction.
15	L	10	30	.....	Atmospheric.	1.50	20	17.81	7.83	43.96
	S	10	25		2 lbs.	1.50	16.66	17.21	7.45	43.28
16	L	10	38	35.60	Atmospheric.	2.33	16.28	21.98	8.50	38.64
	S	10	38		2 lbs.	2.33	16.28	22.55	8.50	37.69

An inspection of these two results still further indicates that increased pressure, *per se*, is an exciting factor. In experiment 15, the reduction in the small tube, with only $\frac{5}{6}$ the amount of gas, was within 0.6 per cent. of that in the large, while the average result of the first 7 experiments showed a difference between the large and small tubes of about 2 per cent. with equal amounts of gas used. Experiment 16 speaks for itself. A careful and unbiased examination of all results obtained, will, I think, justify the conclusion that the effect of increased velocity by itself (other things being equal) on the reduction of iron-ore by blast-furnace gas, at the temperature of melting zinc, is inappreciable, or at least undeterminable with the apparatus used; while increased tension, other things being equal, exerts an exciting influence and enhances reduction, presumably by promoting the transpiration of the gases; thus tending to confirm with the experiments of Mr. Bell, before alluded to, the explanation of Mr. Walsh, rather than that of the author himself.

As regards carbon deposition and its ratio to reduction, the figures seem to indicate that the greater the reduction, the greater the deposition; but so extremely irregular is the action that it can hardly be formulated.

That a portion of these results are to a certain extent contradictory, I am aware; but the mass of their testimony tends to support the correctness of the views above expressed. If these are true, though the figures cannot be held as conclusive, they at least strongly support the conclusion that in a furnace of contracted stack and throat, the increased rapidity due to the constricted area, with which the stock passes through the reducing zone is partially, but by no means fully, compensated for by the acceleration of reduction due to the greater tension of gases.

Though the ideas of Mr. Walsh may be approximated in some such experimental manner as the above, their practicability can of course never be confirmed, and, perhaps, never conclusively denied, till some one has the—may I say?—financial strength of his convictions.

THE MINERALS OF ONTARIO AND THEIR DEVELOPMENT.

BY WILLIAM HAMILTON MERRITT, F.G.S., TORONTO, CANADA.

(Buffalo Meeting, October, 1888.)

A BRIEF paper on this subject (which might readily be made to fill a volume) is suggested at this time by several considerations, among which may be named the meeting of the Institute on the borders of Ontario, not far from some of the mining undertakings which are hereafter mentioned; the mineral developments made in the Province quite recently; the highly creditable collection of most of the minerals of Ontario, for the first time got together, and exhibited this year at Cincinnati, in the Centennial Exhibition of the Ohio Valley; and, finally, the recent appointment, by the Government of Ontario, of a Commission to inquire into, and report upon the mineral resources of the Province, and the best means to be adopted for their development.

The Province of Ontario has no Department of Mines or official geologist, to take note, and keep record, of mining and mineralogical development and discovery. Within the last few years some valuable statistics have been collected by Mr. A. Blue, the Deputy Minister of Agriculture; but recently a desire has risen on the part of the people to obtain further knowledge of these subjects, and the government has appointed a Commission, the report of which is expected to be issued in part, if not in completed form, this winter.

With reference to the mineral exhibit at Cincinnati, it is but just to say that the Government of Ontario decided, only at a very late hour, to make a display of the mineral products of the Province. The exhibit, however, is considered to have been very creditable. The number of specimens was over 2000, comprised in 450 separate exhibits. The total weight of about ten tons testifies to the economic character of the specimens.

They were all from different localities, and carried (as the following condensed summary will show) a wide range: Gold (30 exhibits, including gold-quartz, auriferous mispickel and sylvanite); Silver (52 exhibits, including native silver, argentite, brittle silver, animikite, huntilite, tetrahedrite, silver-nickel, argentiferous galena); Copper (39 exhibits, including copper-glance, native copper, copper pyrites, peacock ore and malachite); Iron (61 exhibits, chiefly magnetite; also, hematite, kidney and specular ores, clay-iron-stone and bog-iron-ore); Nickel (6 exhibits of niccoliferous pyrrhotite and arsenical pyrites); Lead (20 exhibits of galena); Zinc (5 exhibits of blende); Arsenic (3 exhibits of mispickel); Plumbago (4 exhibits of graphite); Asbestos (3 exhibits); Actolinite (1 exhibit); Mica (9 exhibits); Granite (3 exhibits); Marble (6 exhibits); Serpentine (2 exhibits); Sandstone (7 exhibits); Limestone (7 exhibits); Brick, Terra-cotta and Clay (18 exhibits); Gypsum (5 exhibits); Lime (4 exhibits); Hydraulic cement (1 exhibit); Marl (3 exhibits); Phosphate of lime (14 exhibits); Baryta (4 exhibits); Salt (16 exhibits); Mineral water (1 exhibit); Petroleum (2 exhibits); Lignite (2 exhibits); Lithographic stone (1 exhibit), and 82 exhibits of mineralogical and lithological specimens.

The boundary of the Province of Ontario has but recently been defined on the west at the Lake of the Woods. It extends thence eastward on the north side of the great lakes to the junction of the Ottawa river with the St. Lawrence, and northward up to the southern end of Hudson's Bay. Most of this enormous area is composed of rocks older than the Silurian.

An area of Silurian and Devonian occurs south of Hudson's Bay; Manitoulin Island, in Lake Huron, is composed of Silurian strata; Silurian or Devonian formations make up the Peninsula of Ontario, north of Lake Erie, and Silurian strata fringe the northern side of Lake Ontario. Further east, a small area of Lower Silurian lies between the St. Lawrence and Ottawa rivers, and east of the Laurentian belt, which runs across into New York State, and causes the formation of the Lake of the Thousand Islands. All the rest of

Ontario is composed of Archæan rocks of the Laurentian and Huronian series, save where the Animikie and copper-bearing or Nepigon series lie above them in small patches on the north and northeastern shores of Lake Superior.

The more these older rocks are investigated, the more clearly is their value recognized as the bearers of useful minerals. Especially is this the case with the Huronian, which occupies great stretches of country, north of Lakes Huron and Superior. These rocks are similar to the great iron-bearing series of northern Michigan and Minnesota, which, in several places in Ontario (as recently near Ishpeming, Michigan), have also been proved to carry gold-bearing quartz-veins.

Large deposits of copper-ore, occurring, like the iron ranges, in strong belts, are being opened up in this Huronian series, as will hereafter be described; and it has long been known that many quartz-veins occur in this formation, carrying copper-ore, similar to those veins which yielded from six to seven million dollars' worth of this ore at the Bruce mines, near Sault St. Marie.

The Laurentian varies considerably in different parts of the country. Large sections of it in Ontario appear to carry practically unlimited amounts of iron, phosphate and mica. Gold is found occurring with mispickel in Hastings County, north of Lake Ontario, in rocks that have generally been considered Laurentian, but the age of which is open to question.

Granites and marbles, of very superior quality, are obtained from both of these Archæan formations.

The most recent mining discoveries in the Province have been made in the vicinity of Sudbury, in Algoma, where copper and gold have been developed in the Huronian series. The rocks here consist of quartzites, chloritic, hornblendic and talcose schists and diorites, both coarse- and fine-grained, the diorites and schists being especially prevalent about Sudbury. The copper occurs in a highly pyroxenic diorite belt, and, as a rule, associated with a niccoliferous pyrrhotite, the average composition of the ore being from 3 to 7 per cent. of copper pyrites, 2½ to 3½ per cent. nickel, about 63 per cent. pyrrhotite, and 30 per cent. rock. Some 10,000 tons have been mined up to the present time. The ore occurs irregularly in large lenses, dipping S.E., and is also found disseminated through the diorite, with which it has undoubtedly been brought up. The belts can be traced for a long distance through the country, and the ore-deposits, at uncertain intervals, are distinguishable by the red color

given to the soil by the decomposed ferruginous matter. The ore is hard, and the ore-lenses, as a rule, form hills or mounds.

Examples are seen (as at the Vermilion copper shaft), where chalcopyrite forms veins and masses, with breccia of diorite, associated with which native copper has been found in leaf-form. It is possible that this rich copper-ore is due to a secondary action, and has leached or segregated out from the diorite. Free gold has been found with the diorite in which the copper occurs. Rich specimens of gold are obtained in this district from a quartz-vein, cutting a close-grained dioritic rock.

The Port Arthur district, already known to be productive in silver, through the Silver Islet mine,—which yielded some three million dollars' worth of silver-ore,—has recently been developed in a new locality, west and southwest (Silver Islet is southeast) of Port Arthur. The silver-bearing veins cut the black shales of the Animikie formation, which rest unconformably in a horizontal position on the Huronian, and occupy an area in this district estimated at about 1500 square miles. In places they are capped by a thick overflow of trap. A great number of veins occur in this district, cutting through the above-mentioned series so far as followed. They generally occupy faults which have thrown the beds on one side of the vein above or below those on the other side. Sometimes these veins cross intrusive, vertical dikes, as at Silver Islet. The capping of trap has prevented the weathering-down of the underlying shale, so that, at the escarpment formed where the overlying trap occurs, it is possible to see the veins.

The common gangue-minerals of the veins, to the southwest of Port Arthur, are quartz, amethyst, calcspar, fluorspar and heavy spar. They carry iron pyrites, galena, zinc blende (often rich in silver), with native silver near the surface, and argentite below. The galena, is, as a rule, not argentiferous.

The veins run in different directions—N. and S., N.E. and S.W., N.W. and S.E., and E. and W., the latter being the strike of the majority. Neither direction appears to possess any advantage over the others. Of the richest developed thus far, the Silver Islet strikes N. and S., and the Beaver and the Badger N.E. and S.W. Three mines in this district (southwest of Port Arthur), though only in process of opening, have produced about half a million dollars' worth of silver thus far, and there are some fourteen other locations, all of which have been proved to have ore of value.

It may be said, in short, that both of the above-mentioned dis-

tricts of Sudbury and Port Arthur have, for the work done in them, given results in the highest degree encouraging; and it is altogether probable that both will prove to be important mining centers.

Among other new developments worthy of mention is the beautiful banded marble of the Huronian series, which is now being developed by a Chicago firm near Garden river in Algoma. This marble occurs in a wide belt, running a little north of east, and crops out in the side of a mountain, where it is quarried in an open cut. It is beautifully tinted in shades of white, green and pink, is exceptionally hard, and takes an excellent polish.

A development of Laurentian marbles has lately taken place near Madoc and Bridgewater in Hastings county. The quarries are just getting deep enough to turn out very fine merchantable marble. The colors are white, pink, and a dark gray, which polishes almost black. Great quantities of these marbles occur in the Laurentian formation north of Lake Ontario, similar to those which have been so successfully developed at Gouverneur in New York State. One band has been worked for years at Arnprior, and another quarry is being opened at Renfrew. The limited market alone prevents their extensive development.

Another new industry near Buffalo may be worthy of mention. It is the establishment of calcining-works at the white-gypsum deposits, on the Grand river in Haldimand county, to manufacture plaster of Paris. At present the quarries in Nova Scotia and New Brunswick provide most of the material used for this purpose, both in the United States and Canada.

Finally, I would mention that beds of superior clay for pressed bricks have been developed near Hamilton City, at Milton, and at Deseronto. Excellent terra-cotta for interior and exterior use in dwellings was exhibited at Cincinnati, made from the former; superior pressed brick of a dark-red color is made at Milton; and at Deseronto a great variety of patent porous bricks and tiles is manufactured for use in fire-proof buildings.

The statistics of the mineral production of the Province are not obtainable from any official source, except where they are alluded to in the Report of the Geological Survey of Canada relating to statistics of mines and minerals. This report, though only in its second year of issue, is beginning to meet a great want on the part of the public, and especially of mining men and capitalists; it was badly needed, and it is a valuable addition to the Survey.

From figures obtained from the Survey Report and elsewhere,

both before and since its publication, the following statement has been compiled. It represents approximately the production of the Province of Ontario for the twelve months preceding this meeting of the Institute.

	Quantity.	Value.
Arsenic,	120 tons.	$1,200
Fibrous serpentine,	800 tons.	6,000
Copper,	400,000 lbs. (in ore.)	42,000
Gold,	450 oz.	6,760
Gypsum,	5000 tons.	7,500
Iron ore (from July 1st, 1887, to October 1st, 1888),	17,143 tons.	49,906
Mica,	20,000 lbs.	27,200
Nickel,	200,000 lbs. (in ore.)	65,000
Petroleum,	763,933 bbls.	593,868
Phosphates,	3000 tons.	45,000
Salt,	375,000 bbls. (280 lbs.)	140,000
Silver,	383,630 oz.	368,284
Granite, Marble, Building stone, Lime, Cement, Bricks and Tiles,	.....	2,153,084
		$3,505,802

These minerals are produced in the following places, and occur in the formations designated:

Arsenic.—Produced from the arsenical gold-veins in Hastings county, occurring in the Laurentian formation.

Fibrous Serpentine.—In the Laurentian formation in Hastings county; in veins and irregular deposits at junction of dolomite and gneiss. It is ground up at Bridgewater, and used for fire-proof roofing.

Copper.—Found as copper pyrites at a number of places in the Huronian, especially in the vicinity of Sudbury, and also in the Lake Superior district; also, as native copper, in the copper-bearing series at Michipicoten Island and Mamasse Point, Lake Superior. The production mentioned is from the former place; but the Michipicoten mine is being re-opened.

Gold.—In many places in the Huronian series; in the Lake of the Woods; at points between Lake of the Woods and Port Arthur; at Jack Fish Lake in a large vein as sylvanite; from Goulais Bay, northwest of Sault St. Marie; at several places between Sault St. Marie and Sudbury to the east—notably at the Vermilion mining property in Denison township. Very rich specimens of native gold are obtained from the Lake of the Woods and Denison. Gold is at present extracted to a small extent from mispickel veins in Hastings

county. The mispickel carries in places about $14 per ton of gold, and there are great quantities of it. It has been obtained containing 6 ozs. of gold per ton. Free gold has been found to the north of Napanee, in Addington county.

Gypsum.—Produced on the Grand river, Haldimand county, within 40 miles of Buffalo, in the Onondaga formation, and in this locality is white. Deposits of bluish or grayish color occur higher up the river, at Paris, in Brant.

Iron-Ore.—Magnetite and hæmatite occur in the Huronian series at Gun Flint, Knife and North lakes; also at places from there northeast up to the Kaministiquia river, and again to the east of Port Arthur, being a continuation of the Vermilion, Minn., range to the N.E.; also on the north shore of Lake Huron in different places from Echo lake to Killarney, chiefly as specular ore. In the Laurentian formation, from west of Kinmount, in Victoria county, eastward to Frontenac county, and near the Ottawa river, there are a series of rich iron-ore deposits, chiefly magnetite; but hæmatite is found in places. The production mentioned is from deposits along the Kingston & Pembroke Railroad in Frontenac county. Other mines, opened in the district mentioned, are at present idle, owing to the duty on ores entering the United States and the want of blast-furnaces in the Province. These Ontario Laurentian iron-ores are, as a rule, low in phosphorus; but some of them carry a little sulphur. There are, also, considerable deposits of magnetite containing titanic acid; but these are not worked.

Mica.—From the Laurentian formation in the counties of Frontenac and Lanark.

Marble.—Worked in the Laurentian at Arnprior on the Ottawa river, and in Hastings county. It occurs in many other places. It is worked in the Huronian series at Garden river, Algoma.

Petroleum.—Extracted from the Devonian formation in the western part of the peninsula of Ontario. The present production comes from the county of Lambton, from the vicinity of Petrolia and Oil Springs. This petroleum resembles that from Ohio. It has more sulphur and heavy oils than the Pennsylvanian. Much ingenuity is displayed in the refining processes, and the best quality of coal-oil is produced by some of the refiners.

Phosphates.—Apatite occurs extensively in the Laurentian formation in eastern Ontario, in beds or deposits and in veins, chiefly in the counties of Lanark and Frontenac, and, as a rule, in a pyroxene rock associated with hornblende, calcspar and scapolite.

Salt.—Produced by pumping and evaporating brine from thick beds of salt which occur in the Onondaga formation. The manufacturing is most largely developed in the counties of Huron, Bruce and Lambton, in the western part of the peninsula of Ontario. The extent of the beds is very great, and the quality of the brine is excellent.

Silver.—Produced from veins occurring in the Animikie black slates or shales to the southwest of Port Arthur, in Algoma. Found with lead in many veins in the Huronian; but none of them have been worked successfully as yet.

Granites, building-stones, limestones and brick-clays are found in many places in the province in large quantities.

Cement is made from a bed in the Clinton formation, at Thorold in Lincoln county, and from a bed, probably of the Trenton formation, at Napanee Mills, in Lennox county.

WATER-GAS AS A STEAM-BOILER FUEL.

BY D. S. JACOBUS, HOBOKEN, N. J.

(Buffalo Meeting, October, 1888.)

IT is proposed to give in this paper an estimate of the cost at which carburetted and uncarburetted water-gas will have to be sold, in order to compete successfully for steam-boiler use with anthracite coal.

The following are analyses of the gas direct from the generator, and of the same after it has been carburetted for illuminating purposes.

	Per cent. by Volumes.	
	I. Uncarburetted.	II. Carburetted.
Nitrogen,	4.69	2.5
Carbonic acid,	3.47	.3
Ethylene,	0	12.5
Oxygen,	0	.2
Benzole vapor,	0	1.5
Carbonic oxide,	36.80	29.0
Marsh-gas,	2.16	24.0
Hydrogen,	52.88	30.0
	100.00	100.0

Analysis I. is taken from a report on the Granger water-gas, by Dr. G. E. Moore;* II. is an average of a number of analyses of an illuminating gas made by a similar method.

By reducing the above analyses to percentages by weight, we obtain the results given in Table I.

In the use of this gas as a steam-boiler fuel, the products of combustion are not reduced enough in temperature to allow the water-vapor contained therein to condense and thus part with its latent heat. In calculating the calorific power of the gas, the heat that is

TABLE I.

	Uncarburetted.				Carburetted.			
	Per cent. by Volume.	Density by Experiment.	Weight.	Per cent. by Weight.	Per cent. by Volume.	Density by Experiment.	Weight.	Per cent. by Weight.
Nitrogen	4.69	0.972	0.045587	9.03	2.5	0.972	0.02430	3.84
Carbonic acid	3.47	1.524	0.052883	10.47	.3	1.524	0.00457	.72
Oxygen	0				.2	1.1056	0.00221	.34
Ethylene	0				12.5	0.978	0.12225	19.35
Benzole vapor	0				1.5	2.770	0.04155	6.58
Carbonic oxide	36.80	0.973	0.358064	70.89	29.0	0.973	0.28217	44.66
Marsh gas	2.16	0.559	0.012074	2.39	24.0	0.559	0.13416	21.23
Hydrogen	52.38	0.069	0.036487	7.22	30.0	0.069	0.02070	3.28
Total	100.00		0.505095	100.00	100.0		0 63191	100.00
	Calculated density, 0.505 times that of air.				Calculated density, 0.632 times that of air.			

imparted to the water-vapor will therefore be subtracted, in order to determine the amount that is available for heating purposes.

It is also assumed that the heat imparted to the gas during the process of its manufacture is lost by radiation and conduction, this being the case when the gas is stored in a holder and transmitted to the consumer by means of a system of piping.

* Mr. Emerson McMillin, in an article on fuel-gas, presented to the American Gas-Light Association, at its meeting held in New York, October 20, 1887, gives an analysis of uncarburetted water-gas, having the following constituents in percentage by volume; N, 2.0; CO_2, 4.0; CO, 45.0; CH_4, 2.0; H, 45.0; O, 0.5; water-vapor, 1.5. He adds, however, that since calculating the tables used in his paper, he is satisfied that this sample of water gas is not an average one, the CO being too high and the H too low. It would therefore appear that the analysis I have adopted is more nearly an average one.

The calorific power of the gases in British heat-units is given in Table II.*

TABLE II.

	Uncarburetted.			Carburetted.		
	Per cent. by weight.	Heat-units per pound.	Heat-units in portion contained in one pound of the mixture.	Per cent. by weight.	Heat-units per pound.	Heat-units in portion contained in one pound of the mixture.
Nitrogen	9.03	0	.0	3.84	0	
Carbonic acid	10.47	0	.0	.72	0	
Oxygen	0	0	.0	.34	0	
Ethylene	0	20135	.0	19.35	20125	3896.1
Benzole vapor	0	17847	.0	6.58	17847	1174.3
Carbonic oxide	70.89	4396	3116.3	44.66	4396	1963.3
Marsh gas	2.39	21593	516.1	21.23	21593	4584.2
Hydrogen	7.22	51804	3740.2	3.28	51804	1699.2
Total	100.00		7372.6	100.00		13317.1

Experiments were made to verify the ratio, indicated by Table II., between the heats of combustion of the gas before and after being carburetted, by determining the time and quantity of gas required to evaporate a given weight of water contained in an open vessel, heated by means of a gas-stove. The burner of the stove used for burning the carburetted gas caused air to be mingled with the gas before the latter was burned, thus producing a colorless flame. For the uncarburetted gas, the burner was of the Argand pattern, and air was not mingled with the gas before it was burned. The results of the tests are given in Table III.

From Table III. we obtain an average evaporation, per 1000 cubic feet of uncarburetted gas, of 130.9 pounds of water from the temperature of 212° and at atmospheric pressure, and for the carburetted gas an average of 242.1 pounds. That is, the experiments tend to

* The heats of combustion of the several gases, as given in Table II., are those deduced by Mr. G. E. Moore from the experimental results of J. Thomsen. Since completing the calculations given in this article, I have made the necessary transformations in Thomsen's original experimental results, and find that my figures agree with those obtained by Mr. Moore to within the limits of practical error. As there is no record of the values of the atomic weights employed, it is probable that the variations which occur are caused by our having employed slightly different values for the atomic weights.

show that the ratio of the calorific power of the uncarburetted to that of the carburetted gas, by volume, is $\frac{130\,9}{242\,1} = 0.54$.

The volume of one pound of air at 62° Fahr. is 13.14 cubic feet, hence the calorific powers of the gases per cubic foot on the basis of Table II., should be:

$$\text{Uncarburetted, } \frac{7373 \times .505}{13.14} = 283.3 \text{ heat units,}$$

$$\text{Carburetted, } \frac{13{,}317 \times .632}{13.14} = 640.5 \text{ heat units.}$$

The calorific powers calculated from the analyses of the gases are therefore in the ratio of $\frac{283.3}{640.5} = 0.44$, a figure that is lower than that given by experiment. It is very probable, however, that the effi-

TABLE III.

Gas.	Weight of water evaporated.	Temperature of water at start.	Temperature of room in which experiments were made.	Pressure of gas at meter in inches of water.	Time required to heat the water to its boiling point.	Time required to heat and to completely evaporate the four pounds of water.	Number of cubic feet of gas burned.	Amount of water that would be evaporated from and at 212 Degs. F. by the amount of gas burned.	
								Actual.	Per 1000 cu. ft. of gas.
	Lbs.	Deg. F	Deg. F.	Inches.	Minutes.	Min'ts.	Cu. Ft.	Lbs.	Lbs.
Uncarburetted.	4	56	61	4	12	79½	36.1	4.64	128.5
"	4	59	62	4	11¾	79½	34.8	4.64	133.3
Carburetted.	4	56	61	5½	8½	69	19.4	4 64	239.2
"	4	59	62	5½	10	73½	18.2	4.64	254.9
"	4	68	66	4½	8¼	63	19.8	4.60	232.3

ciency of combustion in the practical tests of the two gases was not precisely the same, and we may therefore conclude that the calorific effects, as calculated from the analysis, are in about the same ratio as would be obtained under maximum efficiency in actually burning the gases. One pound of good Lackawanna coal has a calorific power of 13,000 heat units.* The number of cubic feet of gas

* J. C. Hoadley's Report on Trials of a Warm-Blast Apparatus. *Trans. Am. Soc. of Mech. Eng.*, 1885.

required to produce the same heating effect as is produced in burning one ton of coal will therefore be:

$$\text{Uncarburetted gas, } \frac{13{,}000 \times 2000}{283.3} = 91{,}780.$$

$$\text{Carburetted gas, } \frac{13{,}000 \times 2000}{640.5} = 40{,}590.$$

The efficiency of a boiler will be about the same when burning gas for fuel as when coal is used. If, therefore, we disregard the difference in the cost of attendance, in favor of using gaseous fuel, 40,590 cubic feet of carburetted, or 91,780 cubic feet of uncarburetted gas, must be sold for the same price as one (short) ton of coal, in order to successfully compete with the latter for use under steam-boilers.

We will estimate the probable excess of cost involved in firing with coal, above that of gas-firing. In burning natural gas, it has been found that one man is able to attend to boilers aggregating 1500 H. P., whereas, if coal is used, 200 H. P. will be a fair figure. It therefore requires, in the case of large plants, only about $\frac{2}{15}$ as much labor to fire with gas as is involved in firing with coal. If the wages of firemen are $2.50 per day of twelve hours, the cost of firing one ton of coal in boilers that require 4 pounds of coal per boiler H. P. per hour, will be $\frac{2.5 \times 2000}{12 \times 200 \times 4}$ dollars $= 52.1$ cents.

The cost of firing with gas would be $\frac{2}{15}$ of this amount, so that the extra cost of firing with coal would equal $\frac{13}{15} \times 52.1 = 45$ cents. To make a liberal allowance in favor of the gaseous fuel, we will assume that the extra labor of firing each ton of coal costs 50 cents.

Table IV. contains the prices at which the gas must be sold per 1000 cubic feet in order to be as cheap as coal for boiler use; first,

TABLE IV.

Cost of Coal delivered to boiler-room, per ton of 2000 pounds.	Price of gas per 1000 cubic feet.			
	Not including difference in cost of attendance.		Including difference in cost of attendance.	
	Carburetted.	Uncarburetted.	Carburetted.	Uncarburetted.
$6.00	14.8 cts.	6.5 cts.	16.0 cts.	7.1 cts.
$5.00	12.3 cts.	5.4 cts.	13.6 cts.	6.0 cts.
$4.00	9.9 cts.	4.4 cts.	11.1 cts.	4.9 cts.
$3.00	7.4 cts.	3.3 cts.	8.6 cts.	3.8 cts.
$2.00	4.9 cts.	2.2 cts.	6.2 cts.	2.7 cts.
$1.00	2.5 cts.	1.1 cts.	3.7 cts.	1.6 cts.

disregarding the difference in the cost of attendance, and secondly, assuming that the extra cost of firing with coal is 50 cents per ton.

Conclusions.

(*a*) As the cost of manufacturing the uncarburetted water-gas, by the present processes, not including interest on capital invested, is from 10 to 20 cents per 1000 cubic feet, * it does not appear possible to employ it economically for a boiler-fuel, until considerable reduction in cost of production shall have been made.

(*b*) For culinary purposes, where a small amount of heat is required, and it is of special advantage, to have this heat under ready control, fuel-gas could in all probability be sold at a price that would make it a more desirable fuel than coal. For house-warming purposes, however, that are not of too limited a nature, it is not probable that gas could be sold cheap enough to displace coal; for the efficiency of a good steam-heating plant is about 50 per cent.; and if the heating efficiency of the gas be taken at 80 per cent. on account of any advantage that it might have in applying the heat more directly to the air in the room, the price at which it would have to be sold, in order to be as cheap as coal, could be only three-fifths greater that the figures given in Table III.

(*c*) It would also appear from Table III., that a carburetted gas might possibly be made that would cost less, for an equal heating effect, than an uncarburetted gas, in other words that water-gas might be made a medium for burning liquid petroleum fuels.

THE EQUALIZATION OF LOAD ON WINDING-ENGINES BY THE EMPLOYMENT OF SPIRAL DRUMS.

BY E. M. ROGERS, CENTRAL CITY, COLORADO.

(Buffalo Meeting, October, 1888.)

In hoisting from shafts of considerable depth, the dead weight due to the accumulating length of cable is an important element, and has

* Mr. McMillin, in the article previously cited, recommends the use of a mixed fuel-gas that could be manufactured and stored in the holder, for 7.88 cents, 9.72 cents, and 11.75 cents per 1000 feet with coal at $2, $3, and $4 per net ton respectively, and furnished to the consumer for 25 cents per 1000 cubic feet. This fuel-gas, according to his calculations, would develop, on combustion, 323 heat-units per 1000 cubic feet, or it would possess about ⅓ more heating effect per cubic foot than the water-gas which has been made the subject of discussion in the present paper.

led to the employment of conical drums and other methods of compensation. It is apparent that, while the conical drum may effect an equalization of load at a given depth, it cannot be made to serve the purpose of equalization at all points throughout the journey; and it is a problem of more difficult solution, to determine the curved surface or spiral of a pair of drums, which will insure a perfect balance of ropes and carriage for all positions. Some time since, the writer had occasion to study the form of such a pair of drums, and, thinking his results may be of interest to others, presents the following general solution:

As a general proposition, let it be required to construct a pair of hoisting-drums in such a manner that the system may be balanced throughout, that is, the weight of rope and carriage on one shall balance the weight of rope and carriage on the other for all positions. Since the drums must necessarily be of the same shape and proportions, it is plain that the system may be considered as one drum, winding from the small end, and unwinding from the large end.

Now, if the system is balanced when one drum is entirely wound and the other unwound, and the drums are so constructed that the increment of rope change in weight, together with the corresponding increase or decrease of lever-arm by winding or unwinding, *i.e.*, radius, taken for any particular angular movement, will make the moments constant for all positions of the drum, the system will be balanced throughout.

Let a = radius of small end of drum.

Let a' = radius of large end of drum.

Let k = { weight of skip (which may be considered as additional rope).

Let W = weight of entire length of rope.

Let w = weight per unit (say one foot) of rope.

Then, for the position when the whole rope is suspended from the small end, we have

$$(W + k)a = ka'.$$

After one revolution (k + weight of one turn of rope on large end) $\times$ (radius after one revolution on large end) must be equal to (k + W — weight of one turn of rope on small end) $\times$ (radius after one turn on small end), and also equal to $(W + k)a = ka'$. That is to say, the moments for initial points and for points corresponding to one revolution on either end are, by the hypothesis, equal.

Let us now conceive the drum to be continuous, and suppose it to be unwound to a point at which the radius is unity. Then, by

the hypothesis, the weight of rope and skip for that position must equal the constant moment (say, M), since $M \times$ unity $= M$. Obviously, also, in order that M may be constant, the particular weight hanging at any point, corresponding to any radius, r, must be $\frac{M}{r}$, since $\frac{M}{r} \times r = M$.

Suppose the drum to be unwound to any position, and let the radius at that point $= r$. Then, the weight of rope and skip at that point $= \frac{M}{r}$. Conceive the drum to turn backward and forward from this position, an infinitely small portion of a revolution, corresponding to an arc $= dv$, then the radii will be respectively $r + dr$ and $r - dr$. Let i denote the weight of the increment of rope due to the revolution dv. Since the moment for all positions must $= M$, we have

$$(r + dr)\left(\frac{M}{r} - i\right) = M \quad \text{and} \quad (r - dr)\left(\frac{M}{r} + i\right) = M,$$

whence

$$(r + dr)\left(\frac{M}{r} - i\right) = (r - dr)\left(\frac{M}{r} + i\right). \qquad (1)$$

The expression for differential of arc of a spiral, in terms of radius and arc of revolution, is $dz = \sqrt{dr^2 + r^2 dv^2}$, in which $z =$ arc of spiral, and $v =$ arc of revolution (referred to unit radius).

If we conceive the cross-section of the drum to be a true spiral, that is, a spiral in one plane, the error will be slight, and the weight of increment of rope will be $w\sqrt{dr^2 + r^2 dv^2}$, w being weight of unit of length of rope. Substituting this value in equation (1) we have

$$(r + dr)\left(\frac{M}{r} - w\sqrt{dr^2 + r^2 dv^2}\right) = (r - dr)\left(\frac{M}{r} + w\sqrt{dr^2 + r^2 dv^2}\right)$$

whence

$$M + \frac{M\,dr}{r} - rw\sqrt{dr^2 + r^2 dv^2} - w\,dr\sqrt{dr^2 + r^2 dv^2}$$

$$= M - \frac{M\,dr}{r} + rw\sqrt{dr^2 + r^2\,dv^2} - w\,dr\sqrt{dr^2 + r^2\,dv^2},$$

or by reduction,

$$\frac{M\,dr}{r} = rw\sqrt{dr^2 + r^2\,dv^2}.$$

Squaring,

$$\frac{M^2\,dr^2}{r^2} = r^2 w^2 (dr^2 + r^2\,dv^2) = r^2w^2dr^2 + r^4w^2dv^2,$$

or

$$r^4dv^2 = \left(\frac{M^2}{r^2w^2} - r^2\right) dr^2 \therefore r^2dv = dr\sqrt{\frac{M^2}{r^2w^2} - r^2}; \qquad (2)$$

whence,

$$dv = \frac{dr\sqrt{\frac{M^2}{r^2w^2} - r^2}}{r^2} = \frac{dr\sqrt{\frac{M^2}{w^2} - r^4}}{r^3}, \text{ and}$$

$$\int dv = v = \int d\,\frac{\sqrt{\frac{M^2}{w^2} - r^4}}{r^3}.$$

This may be integrated by parts as follows:

Let $\sqrt{\frac{M^2}{w^2} - r^4} = u$ and $\frac{dr}{r^3} = dz$;

then $v = \int udz = uz - \int zdu$; and

$$z = \int \frac{dr}{r^3} = -\frac{1}{2r^2}.$$

$$du = d\left(\frac{M^2}{w^2} - r^4\right)^{\frac{1}{2}} = \tfrac{1}{2}\left(\frac{M^2}{w^2} - r^4\right)^{-\frac{1}{2}} \times -4r^3dr$$

$$= -2\left(\frac{M^2}{w^2} - r^4\right)^{-\frac{1}{2}} r^3dr = -\frac{2r^3dr}{\sqrt{\frac{M^2}{w^2} - r^4}}.$$

$$uz = -\frac{\sqrt{\frac{M^2}{w^2} - r^4}}{2r^2} \text{ and } zdu = \frac{rdr}{\sqrt{\frac{M^2}{w^2} - r^4}} = \tfrac{1}{2}\frac{d(r^2)}{\sqrt{\frac{M^2}{w^2} - (r^2)}}.$$

$$v = \int udz = uz - \int zdu = -\frac{\sqrt{\frac{M^2}{w^2} - r^4}}{2r^2} - \int \tfrac{1}{2}\frac{d(r^2)}{\sqrt{\frac{M^2}{w^2} - (r^2)^2}}$$

$$= -\frac{\sqrt{\frac{M^2}{w^2} - r^4}}{2r^2} - \tfrac{1}{2}\sin^{-1}\frac{wr^2}{M} + \text{Constant.}$$

$$\text{If } v = \int_{r-a'}^{r-a} dv \text{ then } v \;{}_{r-a'}^{r-a} = \frac{\sqrt{\frac{M^2}{w^2} - a^4}}{2a^2} - \frac{\sqrt{\frac{M^2}{w^2} - a'^4}}{2a'^2}$$

$$+ \tfrac{1}{2}\sin^{-1}\frac{wa^2}{M} - \tfrac{1}{2}\sin^{-1}\frac{wa'^2}{M}. \qquad (3)$$

For ordinary values of a and a', the last two terms will be very small and may be neglected.

By substituting different values for a and a' in equation (3), placing second member $= 2\pi n$, and solving with respect to n (the number of revolutions), a curve may be constructed which will show the values of radius for various numbers of revolutions, and will represent a section of the drum made to fulfil the conditions of the problem.

To find the length of rope for any particular radius, that is, to rectify the curve, we may proceed as follows:

$$dz = \sqrt{dr^2 + r^2\,dv^2}.$$

But from equation (2),

$$dv^2 = \frac{\left(\frac{M^2}{w^2r^2} - r^2\right)}{r^4}dr^2 \therefore r^2dv^2 = \frac{\left(\frac{M^2}{w^2r^2} - r^2\right)}{r^2}dr^2.$$

Then

$$dz = \sqrt{dr^2 + \frac{\left(\frac{M^2}{w^2r^2} - r^2\right)}{r^2}dr^2} = \sqrt{\frac{r^2dr^2 + \left(\frac{M^2}{w^2r^2} - r^2\right)dr^2}{r^2}}$$

$$= \sqrt{\frac{r^2dr^2 + \frac{M^2dr^2}{w^2r^2} - r^2\,dr^2}{r^2}} = \sqrt{\frac{\frac{M^2dr^2}{w^2r^2}}{r^2}} = \sqrt{\frac{M^2dr^2}{w^2r^4}}$$

$$= \frac{Mdr}{wr^2}. \text{ This being the value of } dz, \text{ to find } z:$$

$$= \int dz = \int \frac{Mdr}{wr^2} = \int \frac{Mr^{-2}dr}{w} = -\frac{M}{wr} + \text{Constant.}$$

$$z \;{}_{r-a'}^{r-a} = \int_{r-a'}^{r-a} dz = \frac{M}{wa} - \frac{M}{wa'}; \qquad (4)$$

a result which might naturally be expected, since we have already seen that the weight hanging at any position of the drum $= \frac{M}{r}$, r being the radius for that position.

The application of equation (3) may be illustrated by a special case. For instance:

Let length of rope = 2000 feet.
Let weight per foot of rope = 3 pounds.
Let weight of carriage (or skip) = 4000 pounds.
Let radius of small end of drum = 4 feet.
Then the constant moment $M = 40{,}000$.
Then radius of large end of drum = 10 feet.

(These particular values are assumed to permit easy substitution in the formula.)

Omitting the last two terms, equation (3) becomes

$$v = 2\pi n = 6.28\, n = \frac{\sqrt{\frac{M^2}{w^2} - a^4}}{2a^2} - \frac{\sqrt{\frac{M^2}{w^2} - a'^4}}{2a'^2}.$$

Substituting $a' = 10$ and $a = 4$ in the above, and solving we find $n = 55.8$. That is, the number of revolutions of drum between the initial point and largest radius = 55.8.
For $a = 4$ and $a' = 9$, $n = 53.3$.

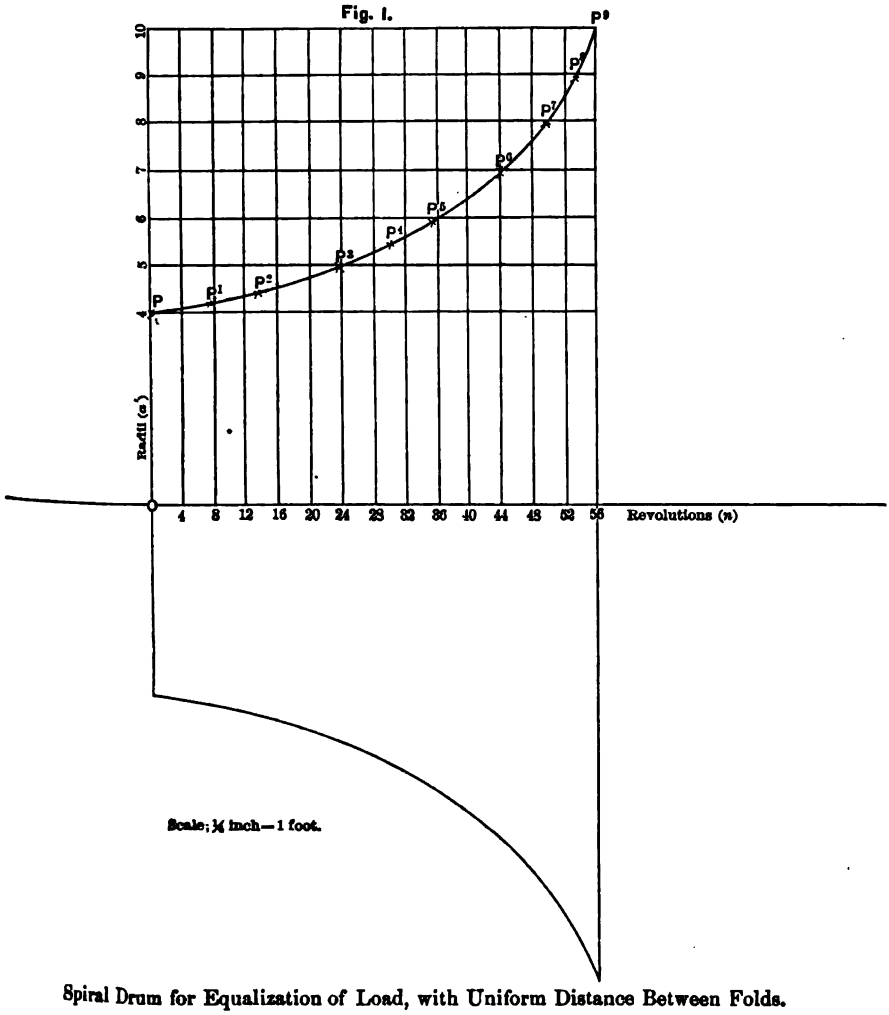
In like manner, by substituting different values of a', from 4 to 10, the following results will be obtained:

For	$a' = 10$,	$n = 55.8$	corresponding to	P^8	on Figs. 1 and 2.
"	$a' = 9$,	$n = 53.3$	"	P^7	" "
"	$a' = 8$,	$n = 49.5$	"	P^6	" "
"	$a' = 7$,	$n = 44.4$	"	P^5	" "
"	$a' = 6$,	$n = 35.1$	"	P^4	" "
"	$a' = 5\frac{1}{2}$,	$n = 30.8$	"	P^3	" "
"	$a' = 5$,	$n = 23.9$	"	P^2	" "
"	$a' = 4\frac{1}{2}$,	$n = 14$	"	P^1	" "
"	$a' = 4\frac{1}{4}$,	$n = 7.3$	"	P^1	" "
"	$a' = a = 4$,	$n = 0$	"	P	" "

Strictly speaking, the construction of the foregoing would give a spiral (in one plane), n being the number of revolutions, or the angular movements, corresponding to different values of the radius a'. However, as before stated, we may conceive the spiral to wind around a drum of constantly-increasing radius, with but slight error, and the construction may be made to show a section of the required drum. Thus:

In Fig. 1, from O lay off horizontal distances corresponding to the movement of rope along the drum for the number of revolu-

tions. In this case, the rope is assumed to be $1\frac{1}{2}$ inches in diameter, and a space of $\frac{1}{2}$ inch is allowed for the ridge between the folds of the rope, making 2 inches the space travelled along the drum for each turn. Also, lay off from O vertical distances, corresponding to the radii, from 4 feet to 10 feet. Then the points, P, P^1, P^2, etc., are points on the curve of the drum determined by assuming certain



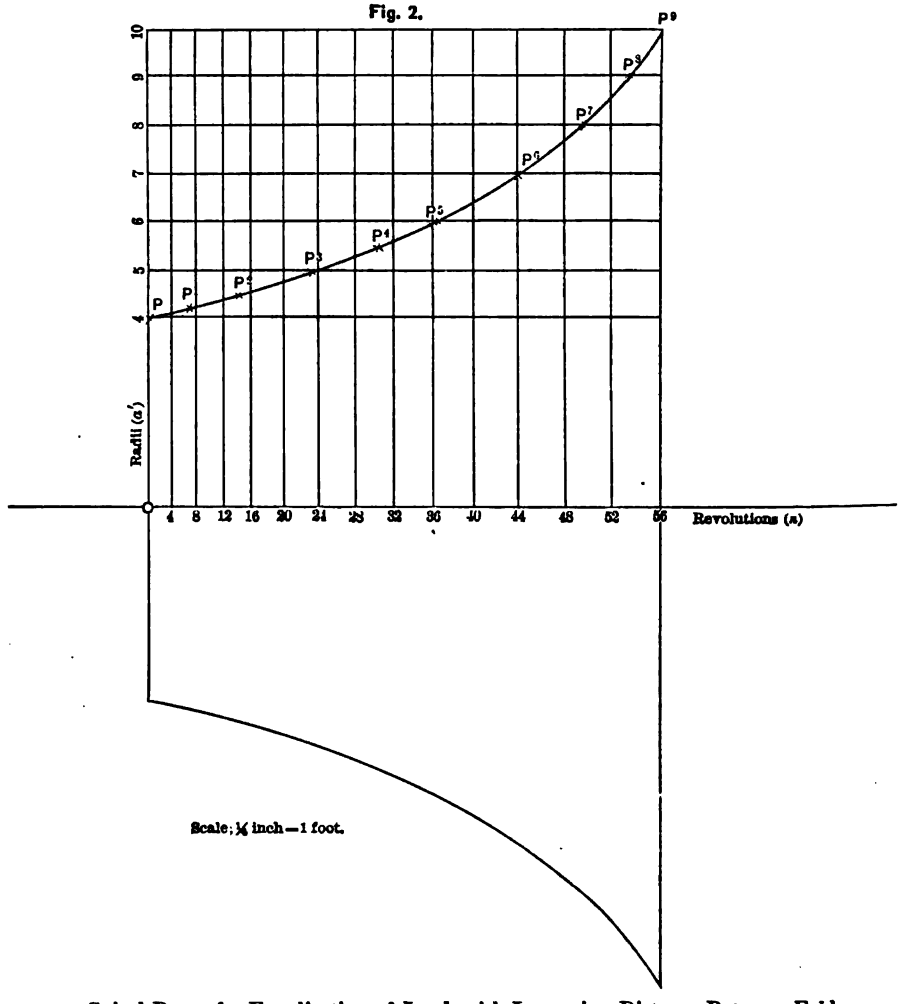
Spiral Drum for Equalization of Load, with Uniform Distance Between Folds.

radii (or values for a'), and finding the corresponding number of revolutions, n. For instance, P^2 is the point at which $a' = 4\frac{1}{2}$ ft., and $n = 14$; P^6 corresponds to $a' = 6$, and $n = 35$; etc.

If the distance between the folds of the rope is supposed to be uniformly increased, the surface will show less curvature, the limit being a cone, with the folds of the rope spread considerably at the large end.

Fig. 2 shows the surface of a drum, with same conditions as in

Fig. 2.



Spiral Drum for Equalization of Load, with Increasing Distance Between Folds from Small End.

Fig. 1, except that the folds are supposed to be close together at the small end, and the distance between them to increase uniformly at the rate of $\frac{1}{16}$ inch for each turn, the object being, not only to de-

crease the curvature, but to pack the rope closely on the straighter portion of the drum, and give room for parting flange and valley on the steeper curve.

The drum must, necessarily, contain the exact length of rope, since, from equation (4), we have the length of rope

$$z_{r-a'}^{r-a} = \frac{M}{Wa} - \frac{M}{Wa'} = \frac{40{,}000}{3 \times 4} - \frac{40{,}000}{3 \times 10} = 2000 \text{ ft.}$$

Again, there must be a perfect balance at all points. For instance, let $a' = 5$, then the length of rope unwound to that radius (Z) = 1333 feet; its weight = $3 \times 1333 = 4000$ pounds; and the moment = (4000 + weight of skip) $\times 5 = 8000 \times 5 = 40{,}000 = M$.

It is apparent that the difference between the radii of the ends of the drum will be less as the weight of carriage or skip increases, and since it is balanced, its weight is limited only by the strength of cable.

In closing, it may be pertinent to remark that it is not the purpose of this paper to comment on the practicability of drums arranged to obtain a perfect balance of ropes, or to discuss the economy of such arrangements, but simply to obtain a general expression from which the form of curve to fit any particular case may be determined.

THE CHLORINATION OF LOW-GRADE AURIFEROUS SULPHIDES.

BY WM. B. PHILLIPS, CHAPEL HILL, N. C.

(Buffalo Meeting, October, 1888.)

IT would be hard to find a mineral region that has been more beset with "processes" for the extraction of gold from auriferous sulphides than North Carolina. And it would be hard to find a mineral region less sought by capital. So many promising mines have come to grief here (perhaps because they were *too* "promising"), so many "processes" have been tried and abandoned, so much expensive machinery has been bought and thrown aside, that to invite money here for investing in gold mines is almost sure to cause a smile of incredulity, or a sigh of disappointed hopes. Within the writer's personal knowledge, at least five disastrous failures have followed the introduction of as many processes. It would seem as

if the very presiding genius of failure had pre-empted the State, and marked it for his own.

It would be a curious question to estimate where the State stands as regards out-go and income in gold mining. Putting the total production of the State, in gold, to date, at $20,000,000 (a very liberal estimate), it is quite probable that she is still on the wrong side of the ledger. The reason assigned for mining failures here, viz., the refractory nature of the ores, is well enough in its way, but does not strike at the tap-root of the matter. It is the cloak with which would-be inventors have sought to hide the multitude of their sins, but has now become somewhat threadbare. The ores are no more refractory than they are in many well-known and thriving camps. They are sulphides, carrying variable amounts of gold and but little silver. The gangue is for the most part quartz, with now and then a little lime and some baryta, as at the Phœnix Mines, in Cabarrus Co., or spathic iron, as at the old Fentress Mine, in Guilford Co., or a chloritic slate, as at the Gold Hill Mine, in Rowan Co., or a talcose slate, as at the Haile Mine, in Lancaster Co., S. C. Most of them crush readily, and present no insuperable obstacle to the stamp-mill and concentrator. Most of them would be smelting-ores, were lead- or copper-ores at hand. They always carry some free gold, fine and coarse, but contain most of their gold in combination or mechanical admixture with pyrite. They generally contain a little copper, sometimes up to 2 per cent.

The proportion of pyrite to gangue varies, it must be confessed, widely, running from 5 per cent. to 30 per cent. The veins of course vary in width to a marked degree, not only as between different districts, but also in the same mine. The walls are generally firm, requiring but little timbering. In spite of many contrary assertions, a considerable experience in North Carolina gold-mines warrants the opinion that the average assay-value per ton of ore delivered to the stamps is not above $16.00. I cannot say exactly what proportion of this is free gold. This is a variant, and not easily come at, except for individual mines. A "clean-up" of about 50 oz. amalgam per week of 6 days is not uncommon at some mines, using 10 stamps and two 5′ x 4′ plates. At other places 75 oz., and even 90 oz., have been obtained. The amount of free gold caught on the inside and outside plates depends more on the way in which the mill is run, and on the fineness of the gold, than upon any other "accidental" circumstance. Very often indeed, the choice of the battery-screen as

to size affects the yield of free gold. I know of a mine where the substitution of a No. 8 for a No. 6 screen was attended by good results.

It is not, however, my purpose at this time to treat of the methods of milling auriferous sulphides containing some free gold. Every one appreciates the importance of saving as much of this as possible; and there are as many ways of running a mill as there are varieties of ore, and, one might say also, of mill-men. The mill must be run according to the character of the ore; and what suits a quartz gangue will not of necessity suit talcose slate. It is to what leaves the plates, the stamped ore, minus the free gold, that I wish to ask attention. We will take it as granted, that the free gold has been caught either in the battery or on the plates or in the slime-trough, and that what we have to do with is a mixture of gangue and pyrite. The pyrite contains by far the greater part of the gold in the ore, and its saving and treatment are matters of prime importance. It is just here that so many processes have failed; and it is because of such failures that North Carolina has become known as "the process State," and the Man with the Process hastens within her borders, and erects an expensive monument to folly. If this should take the form of a funeral pyre, he would not have many mourners in this part of the world.

The attempt to extract gold from pyrite by first roasting and then treating in a Chilian mill came speedily to grief; and the curious may now see near Charlotte a mighty stack, for some years cold as a stone, but once used to desulphurize pyrite. It did so, beautifully. The way in which the sulphur was driven off was admirable; but the residue positively refused to part with its gold by amalgamation.

An attempt was made to matte the roasted ore, but Mr. Spilsbury has given (*Trans.*, xv, 767) the reasons why this did not succeed.

A very amusing attempt was made, though not indeed on roasted ore, but on gravel containing fine gold, to *force* the mercury into contact with the gold. The mixture of pulp and "quick" was put into wooden cylinders, closed at the bottom, and provided with close-fitting pistons, working in suitable stuffing-boxes. Steam-pressure was brought to bear on the piston, and the mercury was forced, not into contact with the gold, but actually through the pores of the wood, and escaped. This process has not been patented.

The matter was going from bad to worse until about 1879 or 1880, when the Mears chlorination process began to be used at the Phœnix mines in Cabarrus County. The principle of chlorinating

gold-ores was not new, but this method of applying the chlorine to the roasted ore was; and it immediately attracted attention. I visited the works soon after they were started, and was much impressed with the feasibility of the idea. Barrel-chlorination promised well from the start; but the Mears process, while indeed it extracted nearly all the gold, leaving but a small amount in the tailings, was too cumbrous and expensive. The manufacture and storage of the chlorine, the hollow trunnion, the goose-neck and pressure-pump, the acid-chamber within the barrel, were unnecessary, troublesome and expensive. But the idea was excellent, of chlorinating in a lead-lined iron barrel under pressure, turning down at the proper time, filtering and precipitating with ferrous sulphate. The mistake that Mears made was in supposing that artificial pressure, applied from without through the goose-neck and hollow trunnion, was the *sine qua non* of success. Pressure is not a factor in barrel-chlorination, but an accompaniment. It is now generated *within* the barrel by the action of sulphuric acid on bleaching-powder, and is due to the evolution of chlorine gas in intimate contact with the ore. But after all, and in spite of the mistakes and misconceptions which led to its abandonment, the introduction of the Mears chlorination process marks a distinct and very encouraging era in the treatment of auriferous sulphides, poor in silver. It was a step in the right direction. The next step was taken by Adolph Thies, M.E., on his assuming the superintendency of the Phœnix mines in 1881, and led to complete success. One by one he discarded the chlorine-generator, the hollow trunnion and goose-neck, the pressure-pump, and the inner acid-chamber. He proved that pressure applied by means of a pump was not necessary to the success of the method; that the introduction of lime hydrate into the barrel and its conversion there into bleaching-powder by the chlorine forced in was not necessary; that the inner acid-chamber and inner flanges were not necessary; and that the best and cheapest results were attained by mixing the roasted ore with water in the barrel to a thin pulp, and then adding bleaching-powder and sulphuric acid for the generation of chlorine.

The method of chlorination, as now carried on by Mr. Thies at the Phœnix mines and at the Haile mine, deserves a new name. It was by his persevering and careful experiments on a working-scale that the Mears process was freed of the hindrances to economical practice which had hampered its usefulness. The Thies process is not the Mears process, any more than the basic Bessemer

is the acid Bessemer. Mr. Thies deserves the full credit for having rescued the Mears process from impending failure, and for having made of it a different though analogous thing. It is time that this was fully understood. He is as much entitled to the honor of making barrel-chlorination a success as any one can be who makes an unsuccessful thing successful. For seven years he has been working along steadily and quietly at the Phœnix mines, perfecting barrel-chlorination. Last March he took charge of the Haile mine in South Carolina, erected two chlorinators, working four tons of roasted concentrates in twenty-four hours, and has run them steadily for several months. The plant will treat eight tons in twenty-four hours.

The success of this method depends on saving every pound of sulphurets possible, and working clean concentrates. The most important factor in barrel-chlorination is the concentration of the sulphurets. If this be neglected, failure is almost inevitable.

At the Phœnix mines, the ore is an intimate mixture of quartz, heavy spar, pyrite, and a little chalcopyrite. Sometimes the pyrite constitutes 30 per cent. of the ore; sometimes not more than 10 per cent., the average being not far from 15 per cent. The more pyrite the more free gold, and the more chalcopyrite the more gold. The heavy spar also appears to have a favorable influence on the ore.

The works consist of an 8×10 in. Blake crusher, two batteries of stamps of 5 stamps each, two Frue vanning-machines, four roasting furnaces, and two chlorinators. Two recent visits to the mines, of several days each, have enabled me to see for myself what they were doing and how they did it, and I desire to express my grateful acknowledgments to Mr. Thies for the uniform courtesy with which he answered my questions and for his kindness in allowing me the run of the mine and works. I also visited the Haile mine where, as before stated, Mr. Thies has erected chlorinators.

At the Phœnix they are chlorinating one ton of roasted ore per day. About one ton of clean concentrates is obtained from seven tons of ore. The roasting-furnaces are of the revolving-hearth type, each roasting three tons of concentrates in thirty-six hours. At the time of my visit they were sinking a new winze from the 350 to the 425 level, and deepening the shaft, so that the output of ore was reduced, and only one furnace was in use. The following is a brief description of the furnace:

Type: Revolving Hearth, Geared to 15 Revolutions per Minute.

Diameter of hearth,	12′
Depth,	8″
Spring (center of pan to spring of arch),	30″
Furnace-wall, thickness,	14″
Grate-bars,	36″
Fire-box,	6′
Grate-surface, width,	2′
Fire-bridge, height,	2′
Throat, length,	4′
Throat, height,	16″
One working-door on each side, opposite,	8″ × 16″
Reverberatory:	
Length,	14′
Width, inside,	6′
Spring,	2′
Two working-doors on each side,	8″ × 16″
Dust-chamber,	4′ × 3′ × 20′
Spring,	1½′

This furnace roasts one ton of concentrates in twelve hours, removing the sulphur to within 0.25 per cent. The cost is about as follows:

Cost of Roasting One Ton Concentrates.

Half-cord wood, at $1.40,	$0 70
12 hours labor, at 9 cents per hour,	1 08
Motive power,	25
	$2 03

Another estimate makes the cost $2.18; but I believe that it can be done for $2.03. As it requires seven tons of ore to make one ton of concentrates, the cost per ton of ore will be 29 cents. About one and one-third tons raw concentrates yield one ton roasted ore, and as the roasted ore assays $28 per ton, the raw concentrates will not exceed $20 per ton. The raw concentrates from some of the Southern mines much exceed this in value. From my note-book I take the following statement: "Raw ore, 172 tons, yielded raw concentrates 15 tons, assaying $51 per ton." This statement can be relied upon as correct, and it shows that some southern sulphurets are better than others, even from the same mine.*

It is well-known that there is a wide difference in the value of sulphurets. Some are very rich, some are fairly so and some are

* I am informed by the superintendent of this mine that a more recent assay, on another lot of concentrates, gave $68, with but little silver.

very poor. Just what determines this variation is not at present known. It appears to be connected in some way or other with the size of the pyrite crystals, since when these are larger and more well-defined the yield of gold is less. Moreover, pyrite of a peculiar greenish-white color seldom runs well.

At the Phœnix mines the concentrates are obtained from two Frue vanners, yielding together about one ton in twelve hours. Blanket-concentration of the tailings was tried, but gave no economic increase of concentrates. The stamps crush to 36-mesh screens.

Just how low the ratio of sulphurets to gangue can go and still leave a margin of profit is a question upon which I do not now care to enter. Within a short time I hope to offer some figures obtained in actual work. Suffice it to say at present that this depends upon three things: 1st. the value of the sulphurets; 2d. the system of milling; 3d. the degree of concentration.

The chlorination is carried on within lead-lined iron cylinders, 60 × 42 in., revolving on the major axis 20 times per minute, with heads bolted on, and with a discharge-valve, 10 in. in diameter, situated on the middle cross-section. Each cylinder is also provided with a small valve, near the head, which serves to determine the presence of chlorine in excess within the cylinder, without opening the discharge-valve. At first the cylinders had two-inch cleats on the inside, but these are lacking in the newer machines. The trunnions are solid.

The charge per cylinder is:

Roasted ore,	1 ton.
Water,	100–125 gals.
Bleach,	40–50 lbs.
Sulphuric acid, 66° B.,	50–60 lbs.

The amount of bleach and acid varies with the character of the ore, as when copper is present much more must be used than if copper were absent. At the Haile mine* 15 lbs. of bleach and 20 lbs. of acid have been found sufficient. The Phœnix ore contains copper up to 2 per cent., but at times, even there, 25 lbs. bleach and 35 lbs. acid have been sufficient. It has been found that better results follow the introduction of the bleach and acid in at least two separate portions, maintaining the ratio between them.

In from four to eight hours after starting the cylinder, the reaction is complete. During this time frequent examinations are made by the small valve, to see if the chlorine is in excess. It occasion-

* At the Haile Mine they are now (October 14, 1888) using 10 lbs. of bleach and 15 lbs. acid.

ally happens that upon releasing this valve a strong flow of gas, other than chlorine, escapes from within the cylinder. When this is the case, the cylinder is stopped and fresh bleach and acid are added. Pressure within the cylinder does not always mean chlorine gas; and if one merely consulted a pressure-gauge and inferred from it that he had excess of chlorine in the cylinder, grave mistakes might follow. It is absolutely necessary that there be within the cylinder a considerable excess of free chlorine for at least one hour before discharging.

After revolving at the rate of twenty revolutions per minute for four to eight hours, depending on the character of the ore, the cylinder is turned down and discharged upon the filter. This is a lead-lined wooden box, 6 ft. wide, 8 ft. long, and 1½ ft. deep. A false bottom is laid, provided with numerous small outlets for the draining away of the liquor into the stock-tanks. This bottom was formerly made of planks, but is now made of perforated tiles. Upon it is laid a layer of coarse gravel, ½ to 1 in. in size, and 1 to 2 in. deep; upon that a layer of smaller gravel; and so on up to a height of 6 to 7 in., each succeeding layer of gravel being of smaller material than the one underneath it, and the topmost layer being fine sand. Neither at the Phœnix nor at the Haile is the liquor from the cylinder decanted upon the filter. This has not been, as might be inferred from Mr. Spilsbury's paper (*Trans.*, xvi., 361), the practice at either place. When the chlorination is finished, the cylinder is at once turned down, and its entire contents are discharged upon the filter. The cylinder is then washed on to the filter, and is ready for another charge.

The filtering goes on rapidly, from two to three hours being quite sufficient to filter and wash. The amount of wash-water varies from 250 to 300 gals. per ton. The filtrate should be quite clear. It is run into stock-tanks, either with or without lead-lining, and from these into smaller precipitating-tanks, where the gold is thrown down by fresh copperas-solution. It requires from two to three days for the reddish-brown precipitate of metallic gold to settle. If copper be present in sufficient quantities for extraction, the supernatant liquor is run into vats on scrap-iron, and its copper is recovered as cement-copper. Otherwise it is allowed to flow off.

The precipitated gold is collected, washed on a paper filter with warm water, and melted down for bullion. With proper care it should be .990 to .996 fine; but if care be not bestowed upon it by skimming or otherwise, it may not be above .965 to .970.

As to the cost of chlorinating one ton of roasted ore, I would refer

to a paper by Mr. Thies, to appear in the forthcoming volume of the California State Mining Bureau. Such figures as I have obtained lead me to believe that it will not be less than $1.50, and may reach $2. Taking the higher figures, we have for the total cost of roasting and chlorinating *one ton of concentrates:*

Roasting, ...	$2 18
Chlorinating, ...	2 66
	$4 84

This I believe to be the maximum cost; and it is only fair to Mr. Thies to say that he has done this work for 75 cents to $1 less. That it can be done regularly for $3.15, as has been stated (See Mr. Spilbury's paper, previously referred to), I cannot believe. It is of course possible that one or two lots can be worked at this figure; but when we take into consideration the unavoidable interruptions and delays, it is not probable that anything like as low a cost can be attained.

It is claimed by Mr. Thies that a sulphide-ore assaying $10 per ton can be worked at a profit by his chlorination process. From what I have seen of it, I think the claim is well-founded. But it must be clearly understood that this method requires the most careful management, and that those who imagine chlorination to be the universal panacea for such ores, to be applied by almost any one, will surely find themselves disappointed.

Mr. Thies's success at the Phœnix mines seems to me to be due chiefly to two things: first, the system of single-hand drilling, which he inaugurated there, and secondly, high concentration. By the first he has worked a 16-inch vein, and by the second he has reduced the cost of roasting, and the bulk of the material to be chlorinated, while facilitating the operation, and enhancing the efficiency of the latter process. The presence of barite in the ore renders the concentration somewhat less effective, as the specific gravity of pyrite is 4.83 to 5, and of barite 4.3 to 4.72.

In concluding these brief notes on a process which has proved itself to be especially applicable to our auriferous sulphides, there are four practical suggestions that present themselves:

1. Whenever possible use the single-hand system of drilling, thus economizing labor, steel, explosives and timber.

2. Be sure that the system of milling and concentrating suits the ore. At least one very expensive mistake has been made in these parts from neglect of this simple axiom.

3. Roast "*dead*," leaving not above .25 per cent. sulphur in the concentrates.

4. Be sure that excess of chlorine is present in the chlorinator, and wash the filter thoroughly. Otherwise the tailings will assay $4 to $5 per ton, instead of 75 cents to $1.

THE LIFE-HISTORY OF NIAGARA.

BY JULIUS POHLMAN, BUFFALO, N. Y.

(Buffalo Meeting, October, 1886.)

THE history of Niagara Falls, as currently told, is simple, and by that very simplicity it has been rendered plausible. As the story runs, the Falls were once situated at Lewiston, 7 miles to the north of their present site; and, since all the rocks in this section are alternate hard and soft strata, the waters acting readily upon the soft shales caused their removal, and the overlying hard layers, deprived of support, broke down under the weight of the large volume of water that rushed over them; and this erosive action has sent the edge of the fall more and more southward, until it now has excavated the seven-mile gorge between Lewiston and the village of Niagara Falls. (See Fig. 1.)

As early as 1790* we find this theory advanced in a description of this region; and it has been practically endorsed by subsequent observers, the most prominent of whom was Prof. James Hall, the State Geologist of New York, who surveyed the Falls in 1841, and gave the most exact description of their surroundings in the publications of the State Survey.† Not until 1883 was the correctness of the accepted theory of the Falls questioned.‡

The geology of the basins of the Great Lakes, as elucidated by Newberry,§ Spencer,‖ and others, has added largely to a clearer understanding of the life-history of the Falls. The investigators have shown us that Lake Erie, as a lake, is, geologically speaking, quite young; that its valley was excavated in pre-glacial times by a series of rivers; and that the Maumee, Sandusky, Cuyahoga, Alleghany and other rivers carried the drainage of the surrounding

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* Andrew Elliott, *Mass. Magazine*, July, 1790.

† James Hall, *Natural History of N. Y.*, Part IV., p. 383.

‡ Julius Pohlman, *Proc. Am. Ass'n for the Adv. of Science*, 1883, p. 200.

§ J. S. Newberry, *Geology of Ohio*, vol. ii., chap. i.

‖ J. W Spencer, *Proc. of Am. Phil. Soc.*, vol. xix., p. 300.

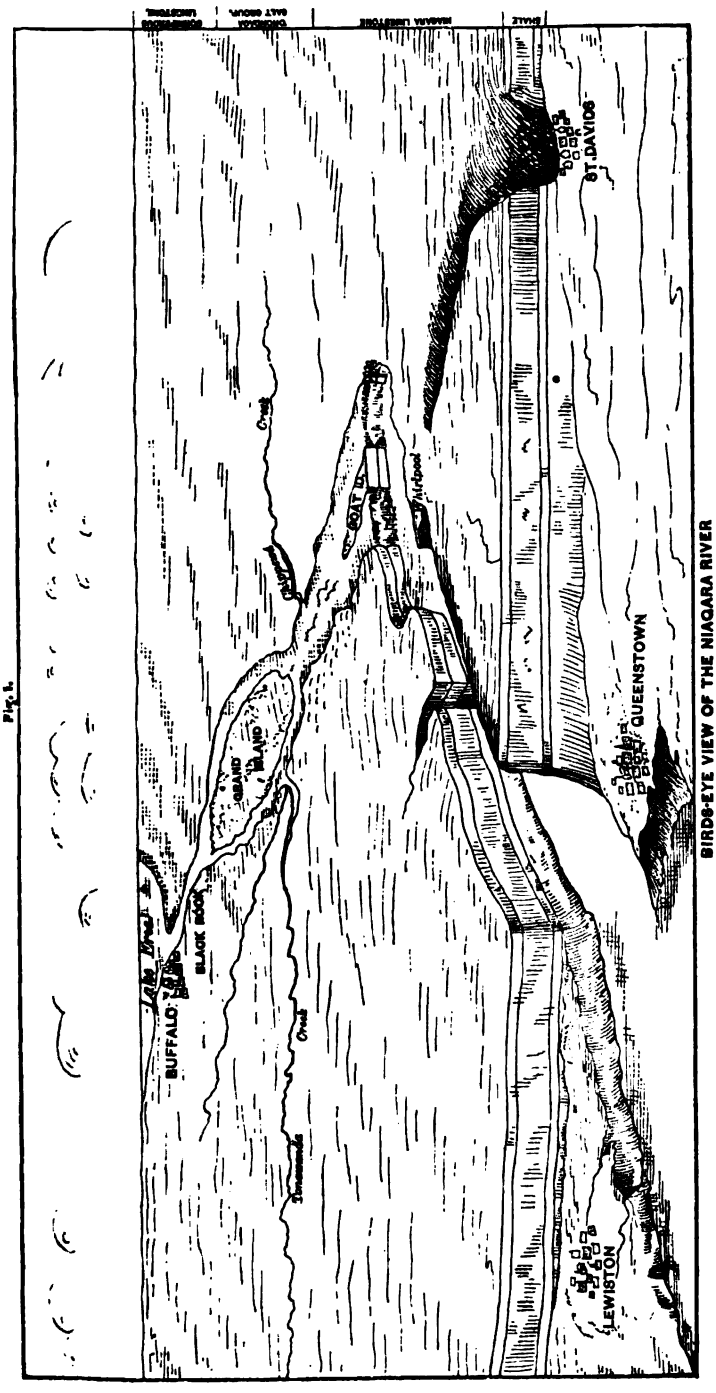
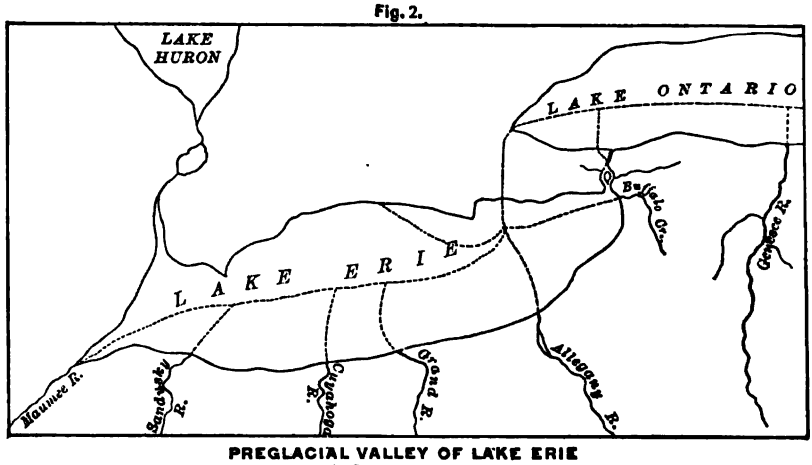


Fig. 1. BIRDS-EYE VIEW OF THE NIAGARA RIVER

lands through Canada, from a point nearly opposite Dunkirk, N. Y., to what is now the city of Hamilton, Ontario, through what has been called the Dundas valley, into the western end of the valley of Lake Ontario, and along its deepest axis through the Mohawk and Hudson river valleys into the Atlantic. (See Fig. 2.)

Even Hall and Lyell have assigned to the largest portion of the Niagara river a pre-glacial age; but, as geology was at that time (1841) practically in its infancy on the American Continent, they were unable to give any very definite boundaries to the area under discussion. Recent investigations have settled this point very clearly. The section of land which concerns us, in the attempt to trace the history of Niagara Falls, is bounded by four well-defined pre-glacial valleys—on the north by the Lewiston escarpment, forming the

Fig. 2.



PREGLACIAL VALLEY OF LAKE ERIE
(—Preglacial Rivers)

southern boundary of the pre-glacial valley of Lake Ontario; on the east by the western watershed of the pre-glacial Genesee river in New York, and on the west by the eastern watershed of the Dundas valley. The question, how far the territory under discussion extended to the south, was not settled in the publications on the geology of the Great Lakes, because the Alleghany river, running past Dunkirk into the basin of Lake Erie, extended the valley only to within 40 miles of its present eastern termination; but it was solved in the summer of 1884, when the Lehigh Valley Railroad commenced the excavation of a series of canals on the so-called Tifft farm, in the southern part of the city of Buffalo. In the course of the construction of these artificial water-ways test-piles were driven

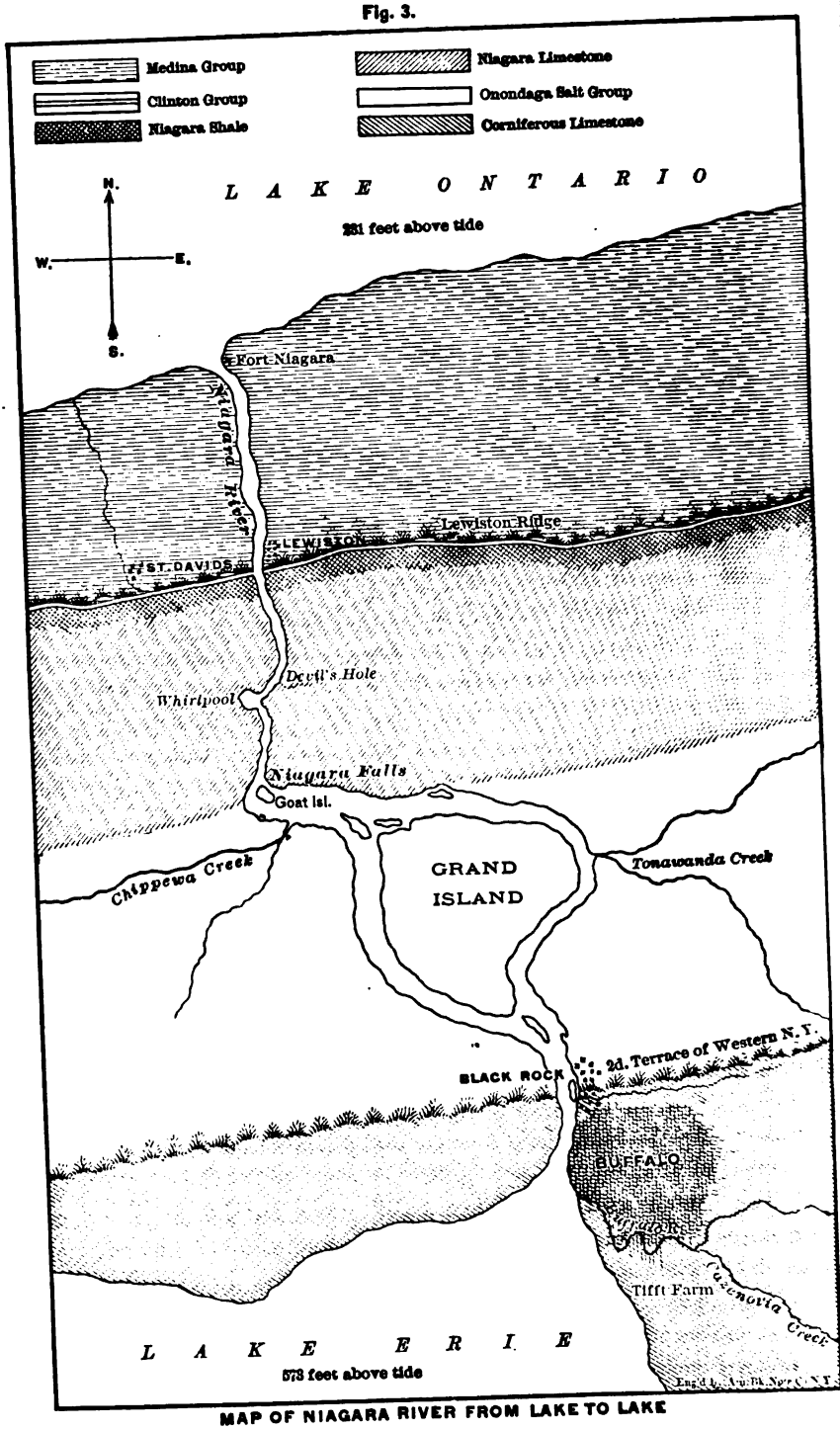
to find out how much blasting would be necessary to excavate the canals to the required depth. Since Buffalo creek flows just to the north of the Tifft farm at a depth of 6 to 10 feet over Corniferous limestone, heavy rock-blasting seemed necessary; but the test-piles revealed the fact that there exists, just to the south of this limestone outcrop, a buried river-gorge, the rock-bottom of which was 86 feet below the surface of the land, or 83 feet below the level of the water, and which ran in a westerly direction into the valley of the lake. The northern edge of this buried river-channel appears in the bed of Buffalo creek, while its southern margin is found on the lake-shore near Bay View, giving a width of the gorge in its upper portion of about 3 miles. The depth of water on the Corniferous limestone, which forms the outlet of Lake Erie into the Niagara river, is from 20 to 24 feet; hence our buried river, flowing 60 feet lower, could not have turned north, but must have continued in a westerly direction to meet the other pre-glacial rivers opposite Dunkirk, making the eastern end of the present Lake Erie the pre-glacial valley of Buffalo creek.

The lake opposite this buried river-channel is in its deepest portion 42 feet deep; and hence we must conclude that the southern and middle parts of the eastern end of Lake Erie are sanded up to the depth of 70 to 40 feet, a fact that agrees well with similar discoveries made by Professor Newberry in Ohio with relation to buried river-channels. The outcrop of Corniferous limestone at the entrance of Niagara river and along the northern margin of Lake Erie gives us the southern boundary of the section in which we have to look for the origin of the falls of Niagara; it also represents the northern watershed of the pre-glacial Buffalo creek, the lowest point of which was evidently in the western part of the city of Buffalo and in the present Niagara river.

From a geological standpoint the area under discussion is extremely simple. Coming from the north we find Niagara limestone from the edge of the Lewiston escarpment to about a mile south of the falls; from there to Black Rock the soft shale and gypsum beds of the Onondaga salt group; and Corniferous limestone over the rest of the area. (See Fig. 3.)

The latter is an almost indestructible stone, and has at present an average altitude in this section of 50 to 60 feet above the level of the lake, whereas the highest ridge of the Niagara limestone is about 50 feet lower. This being the case to-day, it seems probable that the relative altitude of the crests of these two limestone ridges did not vary very materially in pre-glacial times.

Fig. 3.

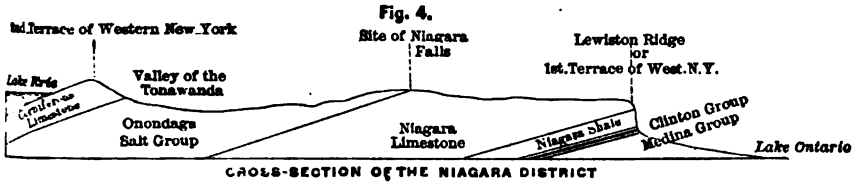


MAP OF NIAGARA RIVER FROM LAKE TO LAKE

Atmospheric agencies, acting upon the soft porous rocks of the Onondaga salt group, necessarily eroded this middle deposit rapidly without making any marked impression upon its limestone boundaries; and soon a line of drainage would be established northerly across the lowest point of the Niagara limestone into the Ontario valley. This erosion produced a shallow depression (see Fig. 4) in an east and west direction, which is now occupied east of the Niagara river by Tonawanda creek, and west by Chippewa creek. It extended, perhaps, 65 miles in an east and west, and 15 miles in a north and south, direction, and deepened as the waters cut their outlet into the Niagara limestone. The erosion of this basin produced the more or less steep face of the second limestone terrace of Western New York, as the erosion of the Ontario valley had formed the first terrace, Lewiston ridge.

There is but one pre-glacial break through the northern margin of the Niagara limestone in this section, and that is at the site of the Falls.

Fig. 4.



CROSS-SECTION OF THE NIAGARA DISTRICT

Here the drainage of the pre-glacial Tonawanda basin cut into the upper thin layers of the limestone on its way to the valley of Ontario, falling over the edge of the Lewiston ridge as one fall, with a height of probably 200 feet.

The action of streams falling over the ridge can be studied to-day in Western New York, in the Genesee river and Oak Orchard creek, and in Canada in the Fourteen-mile creek, flowing past St. Catharine's. In the latter especially we find the action of the pre-glacial Tonawanda well illustrated. A small stream that has collected the drainage from a section of land of the Onondaga salt group has tumbled over the edge of the Lewiston ridge, and from there has been cutting its way southward, forming, by the removal of the soft shale rock between the denser layers of limestone, three falls over these compact strata; the upper and highest over the Niagara limestone; the middle over the Clinton group; and the lowest over the quartzose band of the Medina sandstone. Rapids, which exercise their erosive power upon the shales of the different groups, carry the waters from one fall to the other.

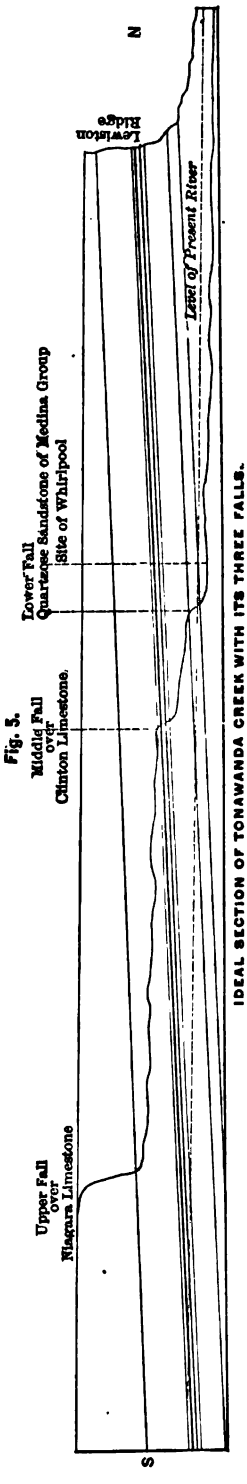

N
Lewiston Ridge
Level of Present River
Lower Fall Quartzose Sandstone of Medina Group Site of Whirlpool
Middle Fall over Clinton Limestone.
Upper Fall over Niagara Limestone
S

Fig. 5. IDEAL SECTION OF TONAWANDA CREEK WITH ITS THREE FALLS.

Restoring the outlet of the pre-glacial Tonawanda according to these models we find it running from the falls almost due north along the present gorge as far as the Whirlpool, and continuing in the same direction into the Ontario valley by the way of the buried channel of St. David's. At the northern margin of the Whirlpool this buried channel is excavated to below the water's edge, *i.e.*, below the quartzose band of the Medina group, which forms the inlet of the river into the pool in the so-called Whirlpool Rapids, and the outlet of the river as it turns, almost at right angles, from the Whirlpool to Lewiston. Hence we must here locate the lowest fall of our pre-glacial creek where it last tumbled over the quartzose band and excavated the underlying Medina shales, after having eroded its gorge from the escarpment to this point. (See Fig. 5.)

All indications as to the exact location of the middle and upper falls have disappeared. We can merely judge by analogy from what is seen in similar streams under similar conditions; and in this way we place the upper fall about $1\frac{1}{2}$ mile south of the lowest fall, which brings it about $\frac{5}{8}$ mile north of the new suspension bridge, or about $1\frac{1}{2}$ mile north of the Horseshoe fall of to-day. The middle fall would then be somewhere a little southerly of the railway suspension bridge.

Restored in this manner, our ancient Tonawanda does not differ in its lower course from streams found under similar conditions at the present time. In its upper course, *i.e.*, while occupying the depression in the shales of the Onondaga salt group, it ran into what is to-day the eastern branch of the Niagara, somewhere near or at the present mouth of Tonawanda creek (see Figs. 1 and 2); then turned south until it met the Corniferous limestone terrace at

Black Rock. Turning north again it formed what is now Grand Island as a peninsula on the American side; here it received the drainage of the western section of this area from Chippewa creek before cutting through the northern margin of the Niagara limestone barrier. The map of the U. S. Lake Survey gives ample evidence of this course in the soundings found in the river.

Cutting its way into the upper thin-bedded strata of the Niagara limestone (in some places deeper than in others) before contracting into one stream on its passage north it left the marks of this erosion in the rapids above the falls, and in the number of larger and smaller islands known to-day as Goat, Luna, Bath, Chapin, Sisters, etc.

This is what we may call the pre-glacial history of the Niagara. The drainage of the shallow Tonawanda valley cuts its way in a northerly direction through the Niagara limestone ridge at the present falls, and outlines and in part excavates the gorge of to-day from the falls to the whirlpool, while its continuation through the St. David's valley is now buried under a deposit of glacial drift.

During the Glacial epoch, the ice-sheet advanced in this vicinity from N.N.E. to S.S.W. This of course, as soon as the Ontario valley was filled with the glacier, closed the northern outlets of all the streams running in that direction, and the Tonawanda, after filling its valley to overflowing, had to find another outlet for its waters. Naturally it turned across the Corniferous limestone terrace near Buffalo into the pre-glacial valley of Buffalo creek, barely two miles away, but at least 120 feet lower. This overflow resulted beyond question in a partial, if not total, destruction of the terrace, thereby opening what is now the outlet of Lake Erie into Niagara river, and determining the ultimate flow of the waters of the Great Lakes in the direction of Niagara Falls, by connecting the pre-glacial valley of the Tonawanda with that of Buffalo creek.

When the Arctic climate again disappeared, the glaciers in their melting not only filled all the valleys to overflowing with water, but deposited also a more or less heavy layer of clays, sands and gravels over the lands they so recently occupied; filling some of the valleys, such as the Dundas, St. David's and Buffalo creek, completely, and others partially, such as the Tonawanda; while the valley of Lake Erie received a deposit of from 60 to 100 feet in thickness.

It is not within the scope of this paper to follow the different drainage-lines of this vast inland ocean resulting from the melting of the ice, or its slow or rapid subsidence with intermediate resting-places where the waters remained stationary long enough to form

beaches along the shore-lines. It is sufficient for our purpose to know that at least Lake Erie and Lake Ontario subsided together, and we take our subject up again when both lakes stood at a level about 30 feet higher than the present Lake Erie; forming two large bodies of water connected by a wide shallow river flowing between Buffalo and Lewiston ridge, which latter existed as a mud-flat at the water's edge, with a clay-deposit on it to the depth of about 30 feet.

Here we meet one of the fundamental questions concerning the Falls: Have they ever been at Lewiston and have they receded southward to their present site from that point? This theory we can safely reject. The Falls, as we see them to-day, or in anything resembling their present form, can never have been at Lewiston. In order to have a *fall*, we must primarily have a difference in level sufficient to form it, and this condition never existed at Lewiston. If we can imagine that Lake Ontario subsided independently, then we have a right to ask what became of the 20,000,000 cubic feet of water which flowed into the completely filled lake-basin every minute? Lake Ontario subsided with many resting-places, as testified by the beaches now apparent; and, with the line of drainage once established in a northerly direction between Buffalo and Lewiston, no power on earth could hold back this immense volume of water and prevent it from cutting its way down into the clay-deposit that covered Lewiston ridge, as soon as the waters of Lake Ontario had subsided ever so little below the edge of the escarpment.

We must assume that the erosion of the clay-deposit occurred just at Lewiston because that happened to be the lowest point in the escarpment; a small streamlet running across the limestone had perhaps pre-determined this place, somewhat in the same manner as the little stream called Bloody Run, on the American side, near the Whirlpool, has produced a large chasm, the "Devil's Hole." Whatever the original cause may have been, the limestone between the Whirlpool and Lewiston was eroded in the form of a shallow valley previous to the ice-period, thus determining not only the point of overflow and the direction of the gorge between Lewiston and the Whirlpool, but aiding greatly in its erosion by reducing the thickness of the upper hard layer of Niagara limestone.

As the gorge of St. David's was filled with glacial drift and covered with a deposit of clay higher than that found at Lewiston, the waters followed the lowest level at the latter point; and after the current had commenced to remove the clay on the top of the ridge, the excavation of a new gorge down into the solid rock was a neces-

sary consequence, even if an already excavated channel existed but a few miles away.

At the time when the waters of Lake Ontario subsided below the edge of Lewiston ridge, the present outlet of Lake Erie into the Niagara at Buffalo was already well-defined in the break of the limestone terrace, and the river was but little broader than it is to-day. From here the waters formed a lake-like expansion covering the lowest portions of the Tonawanda and Chippewa valley, with what is now Grand Island as a mud-flat near the water's level. Near the present site of the Falls, the river contracted again between well defined clay banks, submerging the rapids, all the islands and a part of the site of the present village of Niagara Falls. To about half-way down to the Whirlpool, the river was barely twice its present width; but below that it expanded again considerably, contracting as its outlet at Lewiston deepened by the removal of the clay-deposit. Well-defined terraces on both sides of the river, between the Falls and Lewiston, testify to this part of Niagara's history, and teach us that the waters subsided irregularly.

As soon as the river had removed the clay-deposit from the top of Lewiston ridge, it commenced the work of destruction upon the already partially eroded Niagara limestone along the already-mentioned shallow valley between Lewiston and the Whirlpool. It is possible that a small fall over the twenty feet of limestone existed here for a limited length of time; but as soon as Lake Ontario had subsided sufficiently to expose the underlying Niagara shale, the edge of the limestone would recede very rapidly. With a continued subsidence of the lake, a second fall over the Clinton limestone would come into existence at Lewiston, and finally a third one over the quartzose band of the Medina group; but owing to the vast quantity of water pouring over these falls, and the very marked preponderance of the soft shales over the hard limestones and sandstone, the erosion of the gorge between Lewiston and the Whirlpool must have been very rapid. It seems probable that the gorge was eroded as rapidly as the waters of Lake Ontario subsided, and that in its whole length of three miles there never existed at one time more than one small fall, while the rest presented the appearance of the river as we see it to-day—a long series of rapids.

After the comparatively thin layer of Niagara limestone had been removed along the section between Lewiston and the Whirlpool, another phenomenon appeared. In its southwesterly direction, the newly-eroded gorge met the ancient bed of the Tonawanda at the

Whirlpool, striking it almost at right-angles. Thus, while the Niagara river is of pre-glacial origin from Buffalo to the Whirlpool, the section from the latter point to Lewiston is post-glacial, and its pre-glacial continuation between the Whirlpool and Lake Ontario, through the valley of St. David's, is closed with glacial drift.

Long before the excavation of the lower gorge was completed, the river in its middle course, between the Falls and the Whirlpool, had settled down into this pre-glacial gorge of the Tonawanda, allowing the waters of the lakes to tumble over the Niagara limestone, where the old creek had had its upper fall, about 1½ mile south of its lowest fall near the Whirlpool rapids, or nearly 1½ mile northerly from the present Horseshoe Fall, or where to-day the clay banks on both sides of the gorge show us that the river was confined within limits narrow enough to permit the formation of a fall. Here the present Niagara river formed its upper fall with a height that increased as rapidly as Lake Ontario subsided and the lower gorge was excavated, until at the completion of the latter the lowest fall of the ancient Tonawanda over the quartzose band of the Medina group was left in its old place as the Whirlpool rapids, because its upper edge is below the level of the present river (see Fig. 5); whereas the small middle fall over the Clinton group, travelling southward at a very rapid rate, soon combined with the upper fall, and formed one grand cataract, the first full-born fall of Niagara, with a height of probably nearly 200 feet, at a point perhaps half a mile to the north of the new suspension bridge, *without ever having been at Lewiston at all.* This cataract, having a width of about 1400 feet, naturally assumed a horseshoe form, because some parts of the upper layer of Niagara limestone had been eroded more than others; and here the recession would take place fastest because it held the largest volume of water, and while the underlying shale was more rapidly washed away, the overlying thinner limestone beds would break down more quickly. Of course the deepest notch of the horseshoe may have done then as it is doing in the Horseshoe fall to-day, namely, travel in an irregular manner from side to side. At all events, in a horseshoe form the one fall of Niagara travelled southward until it met the northern apex of Goat Island, where the waters were divided into an American and a Canadian fall.

Goat Island must have extended originally perhaps half a mile further north; and though its northern face is now 1400 feet wide, it must have tapered to a point in pre-glacial times, and this north-

ern extremity was in all probability located nearly opposite Prospect Point, the northern point of the present American fall.

Owing to the fact that the line of the deepest pre-glacial erosion ran along the western shore of Goat Island, and the shallowest (with a large number of islands) was found on the American side, the largest volume of water found its way over the Canadian fall and caused a more rapid rate of recession on that side than could take place on the American side. As a consequence, the whole line of the falls, with the islands, swept in a semi-circle to the south and east with the northern (or, at the beginning, the eastern) point of the American fall as the center, which remained comparatively stationary, owing to the thicker limestone on its top and the limited quantity of water pouring over its edge. The original northern apex of Goat Island disintegrated rapidly; the western end of its northern exposure receded with the rapid recession of the Horseshoe fall, while its northeastern part followed more slowly, with the southern end of the American Fall.

The rate of recession of the Falls and the determination of their age have always been interesting speculations. Various estimates have been given at different times by different observers. They vary as to the rate of recession from one foot per year to practically nothing; and the age that can be assigned to the Falls and gorge varies proportionately. All these computations started with the theory that the Falls commenced their southward movement from Lewiston and excavated the whole seven miles of gorge. Lyell in 1841 considered a recession of one foot per year a fair estimate, and 35,000 years as the probable age. Those who considered this rate of recession as too high only needed to multiply the given age in proportion to obtain even several hundred thousand years. The fact is, that all such calculations were mere assumptions. Nothing definite was known until Prof. Hall made the first trigonometrical survey of this region in 1842. The United States Lake Survey went over the same ground again in 1875; and during the meeting of the American Association for the Advancement of Science in Buffalo in 1886, Prof. R. S. Woodward, of Washington, D.C., again surveyed the edge of the Horseshoe fall for the benefit of the Geological Section's discussion of the Niagara problem. A comparison of these three surveys brings out some surprising facts. The falls recede faster than anybody ever anticipated. In the forty-four years that passed between the first and the third survey, the most southerly or south-

easterly point on the edge of the Horseshoe fall has receded 382 feet, or a fraction less than *nine feet per year*. (See Fig. 6.*)

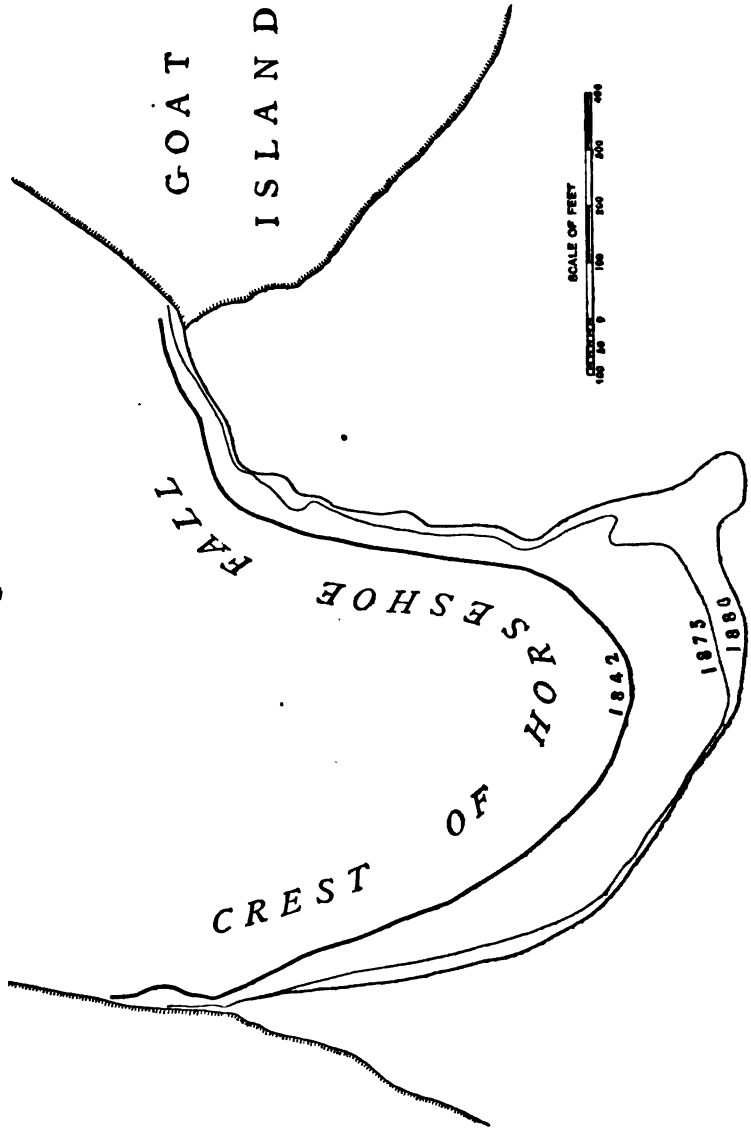


Fig. 6.

As the sides of the Fall do not recede so rapidly, Prof. Woodward computed the total area worn away as follows:

* The original of this tracing was kindly placed at my disposal by Hon. Thos. V. Welch, the Superintendent of the Reservation.

Between 1842 and 1875, 185,000 square feet or 4.25 acres.
Between 1842 and 1886, 245,000 square feet or 5.62 acres.
Between 1875 and 1886, 60,000 square feet or 1.37 acres.

Distributing this over the full length of the contour of the Horseshoe fall, 2300 feet, we obtain the minimum value of the yearly rate of recession as follows:

Between 1842 and 1875 a rate of 2.44 feet per year.
Between 1842 and 1886 a rate of 2.42 feet per year.
Between 1875 and 1886 a rate of 2.38 feet per year.

Upon a minimum average of 2.4 feet per year, the whole face of the fall will travel southward at the rate of one mile in 2200 years. This rate, however, must vary with the increasing thickness of the Niagara limestone, and must have been more rapid further north, decreasing as the Falls receded southerly. This is corroborated by the figures of the survey, viz., 2.44 and 2.38 feet per year. Future surveys will be looked for with interest by all who have studied the problem of Niagara; and if the Commissioners of the Reservation are unable to obtain an appropriation for such a purpose, perhaps the American Association for the Advancement of Science, when it meets again in Buffalo in 1896, will institute a new survey of the Falls for the purpose of verifying the figures obtained in former years.

Aside from its geological interest, the knowledge of the exact rate of this recession has an important bearing upon the question of the age of man on the North American Continent. Whenever reference has been made to the discoveries of implements of human manufacture in Glacial gravels, the age of these deposits in years was computed by what was considered the most exact chronometer of post-glacial times, namely, the gorge of the Niagara. With this as a starting-point, ascribing to it an age of from 35,000 to several hundred thousand years, the age of man on this continent was left somewhat uncertain; and if any geological or other phenomena suggested a smaller number of years for post-glacial time, the gorge of Niagara always presented undisputable evidence of long, long ages. Now at last archæologists and geologists can shake hands across the gorge and rejoice that the great point of contention has been settled to the satisfaction of both.

With the computation, extending over a term of forty-four years, that the minimum average rate of recession of the whole contour of the Falls is 2.4 feet per year, and the knowledge that the Niagara

limestone decreases in thickness as we go north, it seems but fair to assume that 2000 years would be a fair, certainly not too low, figure in time necessary for the Falls to have receded one mile. Beginning the existence of the cataract at a point about 1½ miles northerly from the present site, the age to be assigned to the cataract dwindles down to the quite respectable, but geologically speaking small, number of perhaps 3000 years.

A knowledge of this rapid rate of recession brings, of course, before the eyes of every good Buffalonian the important question: "How long before the Falls will be at Buffalo?" Already the veteran State geologist of New York, Professor Hall, reviews this question, in his report on Niagara Falls, in 1842, as follows: "If present causes continue to operate as now, . . . the consummation of the grand cataract of Niagara . . . . will be either a continuous rapid stream from Lake Erie to Lewiston, or a rapid stream with a low fall at the outlet of Lak Erie."

After the Falls of to-day have receded one mile, there will be again only one fall; the American fall will have disappeared entirely, and the islands on the eastern side will be represented as a series of low hilltops on a peninsula projecting from the American shore. The edge of the fall will then be nearly half a mile to the east of the eastern point of the present Goat Island, with its deepest indentation on the line of deepest water, well over towards the Canadian shore. Owing to the increasing thickness of the Niagara limestone, the edge of the fall would then be about 50 or 60 feet higher than at present, whereas the grade of the river below (15 feet to the mile, down to Lewiston) would bring the waters 15 feet higher up to the base, making the height of the cataract, after it had receded one mile, nearly 200 feet, with about 45 feet of Niagara shale at its base, overlaid with 150 to 160 feet of limestone. All the rocks in this section dip about 20 feet to the mile, from the north to the south, and the upper line of the shale, which is 80 feet above the water at present, would be 20 feet lower after one mile of recession.—But, in this mile, the fall will have ceased already to travel due south, and taken a more southeasterly direction; in the next mile this direction will be more easterly still, with a strong inclination toward the line of deepest water near the Canadian shore. In this direction the height of its upper crest will change but little, as it merely crawls along the line of the limestone outcrop, and not across its dip as heretofore. The actual height of the cataract, however, will be diminished by the rise of 15 feet in the river below, making it in all probably

185 to 180 feet, with an exposure of shale at its base of perhaps 30 feet.

To understand better what will happen after the Fall has receded four miles in all, *i.e.*, when it has reached the northern point of Grand Island, we must go back to the time when the waters of Erie and Ontario stood at the level of Lewiston Heights, and that which is now Grand Island, or, better, the peninsula formed by the bend of the ancient Tonawanda creek (Fig. 3), was a mud flat, partially under water, partially at the water's edge. As the waters subsided, and the land rose, its southern point divided the current of the waters of the lakes as they rushed past the site of Buffalo. The western branch of the river had a free flow down toward Lake Ontario; but the waters which were diverted into the eastern branch found a bed excavated only as far as the mouth of Tonawanda creek, and had to cut an outlet across the clay-deposit of the peninsula in a westerly direction in order to meet again the western branch of the river, forming in this way what is now called Grand Island. With the subsidence of the lake, this cut was deepened; but it is still from 10 to 16 feet shallower than the other parts of the river on both sides of Grand Island.

After the present Falls shall have receded about one mile from their present site, we shall have only one cataract; a further recession of somewhat more than 3 miles will again bring a division into American and Canadian Falls, separated by Grand Island, and as to-day, the largest volume of water will follow the line of deepest excavation, along the Canadian shore. Before this time, the soft shale rock at the base of the fall will be entirely covered by the water of the river below; only solid walls of limestone will be presented to the eroding action of the waters, and the rate of changes will be reduced to a minimum.

As to-day, the Canadian Falls will cut its way backward in a southerly direction *across* the dip of the limestone exposure, while the American Falls of that date will resemble the present one by receding east *along* the outcrops, and for the same reason the latter will be higher than the former.

All the rocks in this vicinity dip, as I have said, about 20 feet to the mile, from north to south; and if, as at present, the river needs a fall of 15 feet to the mile to keep its bed clear, the height of the falls will diminish about 35 feet for every mile which they travel south, and long before they have receded 12 miles, to the southern end of Grand Island, they must disappear entirely as falls,

and the river will present only a long series of rapids cutting into the solid Niagara limestone, because the soft shale of the overlying Onondaga salt group will offer no resistance to the eroding action of the river. The second American Fall, receding around the northern and eastern side of Grand Island, will move more slowly than the Canadian Fall, but will ultimately be reduced to the same condition, forming, between Lake Erie and Lake Ontario, one continuous river with swift-flowing current, and perhaps a few short rapids; one over the outcrops of the Corniferous, now at the inlet of the river, another over the Niagara limestone, and one where to-day are the Whirlpool rapids.

This, of course, presupposes that everything will go on in the future as it has done in the past. But who knows what the ingenuity of men will devise in centuries to come to counteract the natural consequences of running and falling waters? Who can tell what mechanical appliances will be invented to retain the Falls in their present place or to divert the waters of river and lake to some useful purpose in a manner not thought of to-day? Who can tell but that long before the noble cataract has had time to disappear, the chain of the Great Lakes may have suffered the fate of similar large bodies of water in our western territories, which now only give us their beach-lines as indications of their former existence?

THE IMPURITIES OF WATER.

BY A. E. HUNT AND G. H. CLAPP, PITTSBURGH, PA.

(Buffalo Meeting, October, 1888.)

THIS paper constitutes in substance a part of a more elaborate chapter, accompanied with extended tables of analyses, prepared for the book of Mr. Fred. H. Whipple on "Water-Supply." In advance of that use of the material, however, it seemed appropriate to lay a portion of it before the Institute; and this opinion of the authors, the Council, by accepting the paper, has ratified.

It may seem at first glance, nevertheless, that a paper dealing chiefly with this subject in its sanitary aspects, and only treating in a subordinate way of the one subject (boiler-scale) usually deemed to concern our professions, is somewhat out of place in our *Transactions*. But reflection will, we trust, correct this impression. In recent periods, and in continually increasing degree, the managers

of mining and metallurgical operations, especially the latter, have found themselves charged with the care of large numbers of men; and the sanitary conditions of such establishments have become an important factor in their technical and commercial success. A general survey of the organic as well as the inorganic impurities of water may be suggestive and useful to such; and since the chemists at metallurgical works might easily, with slight guidance, and without extra expense to their employers, ascertain the sanitary conditions presented by the water-supply, this subject may well be laid before them. At the same time, it is not proposed that our *Transactions* shall be crowded, to the exclusion of more directly professional material, with details of sanitary science. These must, of course, be sought in special treatises, or at the hands of expert specialists.

To prove the importance of this subject, we need only point to the startling proofs, collected since the announcement of the germ-theory of disease, that a contaminated water-supply is the active agent in spreading numerous epidemics, for which such a cause was formerly unsuspected.

There is, perhaps, no question which more severely taxes the resources of science than the determination of the fitness or unfitness of a water-supply. Not only must waters of bad taste or smell and carrying considerable amounts of foreign matter be rejected, but many waters not only agreeable to the taste, but colorless, odorless, and containing very little suspended matter, are often the abundant sources of disease.

It has long been known that dissolved oxygen plays an important part in the purification of water; that the amount of oxygen contained in water carrying putrefiable organic matter is very low; and that where oxygen is deficient, and organic contamination considerable, fish and higher grades of animal life desert the water, while microscopic examination shows an abundance of the lower forms of vegetable and animal life, such as *algæ* and *infusoria*. It has been further observed that common fish, such as sunfish and suckers, will live nearer the source of contamination than the higher-mettled salmon and bass. Practical data thus given by nature should go hand in hand with the chemical analysis of a water-supply; for the evidence of nature is based upon larger samples, and hence better averages, than the chemist can command for analysis.

Color.—By reason of vegetable and peaty matter, many samples, especially of surface-waters, have a decided color; a yellowish-brown of various shades and degrees of intensity being the prevailing hue.

Professor T. M. Drown has adopted a very satisfactory series of standards for comparison of these shades, by solutions containing the Nessler reagent with certain known amounts of ammonia, so that the degree of color can be expressed in terms of given amounts of Nesslerized ammonia.

These colored waters are not necessarily unwholesome. On the contrary, while they are not as attractive for drinking, and nearly always show more or less albuminoid ammonia, they often contain very little free ammonia, and are sometimes quite wholesome and pure.

Suspended Matter.—In the analysis of drinking-water, the character of the suspended matter, and its capacity to settle, or, on the other hand, to remain in fine suspension, and be served to the consumer through the service-mains, should be the criterion as to the advisability of filtering before analysis. In general, the chemist should seek to include in his analysis whatever the consumer includes in the water he drinks.

Odor.—The odor a water gives should be noted. Many waters have decided odors, some already formed from the gases dissolved in the water, and constantly given off from it, or easily liberated by shaking the water, and sometimes liable to be generated by chemical or other changes in the dissolved or suspended matter. The odors of waters may best be tested in a confined space, as by shaking a wide-mouthed gallon-bottle, half filled with the water, and immediately smelling the air in the bottle, which will be impregnated with the gases from the water.

Some waters give a different odor when cold than when hot; and the odor of some is due entirely to the suspended matter, and can be entirely removed by filtration, while in others filtration does not affect the odor.

Prof. Drown, in his paper on "The Odor and Color of Surface-Waters," read before the New England Water-Works Association, in 1887, suggests a series of terms for describing the various odors of waters, which we believe to be good, and have adopted in our own practice. He says of them:

"'Earthy' is the odor of freshly-turned soil or of a moist hot-house; it often passes into 'musty.' 'Straw-like' is the odor of moist straw; I think no one could mistake it, or fail to recognize it; it is one of the most frequent odors met with in surface-waters. 'Mouldy' is the odor of moist straw which has begun to decompose; there is nothing disagreeable suggested by it; it is also somewhat allied to the earthy or loamy odor. 'Mustiness' is advanced mouldiness and verges on the disagreeable. It is the odor of damp, closed vaults. 'Wooden' is the odor of tubs with hot water; it suggests wash-day. 'Disagreeable' is a more difficult term to

explain, first, because it is not easy to get agreement among different observers on this point, and secondly, because under it we include two classes of odors, namely those of growth and those of decay. Some aquatic plants and low animal organisms give off mal-odorous gases as the product of their normal growth, and all organisms are more or less mal-odorous in the process of decay. Now, it sometimes happens that these two odors cannot be sharply distinguished from one another; they are both unpleasant odors So in the absence of finer discrimination of odors, or the positive identification of the noxious plants or animals, there seems to be no other way than to include them under the head of 'disagreeable.' It is of interest to note in this connection, that river waters rarely give, either cold or hot, these 'disagreeable' odors. They are generally confined to the waters of lakes, ponds, and reservoirs, which have abundant vegetation and but little motion. River waters, even those of considerable organic impurity, give 'straw-like' and 'mouldy' odors, but the unpleasant, fishy, oily and rotting odors are, as far as my experience goes, generally absent. 'Offensive' is a term sufficiently characteristic, it is more than disagreeable and may be reserved for waters undergoing putrefactive change, or for those which contain considerable sewage. Of the odors of the waters at ordinary temperature, the most frequently recurring are 'straw-like' and 'mouldy' and 'disagreeable.' When heated, the 'wooden,' 'earthy' and 'musty' odors are apt to be developed."

"To express degrees of intensity of odor I am in the habit of using the qualifying terms,—'very faint,' 'faint,' 'distinct,' 'decided' and 'strong.' A good deal of personal equation comes in here. What is 'faint' to one is 'distinct' to another and 'very faint' to a third. Still there seems to be need for some degrees of comparison of intensity of odors, and, at all events, the use of any set of terms by the same observer will give comparable results."

"The origin of the odors of surface-waters, when uncontaminated with sewage or other drainage, is clearly referable to the vegetable and peaty matters of the soil over and through which it flows, and to the vegetable or other growths in the water itself."

Water-Analyses.—Sanitary analyses are made (1) to discover, with regard to a given water, whether it contains any organic or inorganic poisons, or such an amount of earthy constituents as will make it deleterious for drinking and cooking; or (2) to determine its value as a mineral (medicinal) water.

Commercial analyses are made to determine concerning a given water, (1) its applicability for making steam; (2) its hardness, or the facility with which it will "form a lather" necessary for washing; or (3) its adaptation to other manufacturing purposes than those mentioned above.

The common impurities of water are held in suspension (organic matter, both living and dead, and silica, clay, or other materials with which the water may have been in contact); or in solution (mineral salts and acids, and the products of the decomposition of organic matter from sewage or other contamination).

The results of water-analyses are reported in grains per Imperial, or United States wine, gallon, or in parts per thousand, hundred-

thousand, or million. The Imperial gallon is reckoned as 277 cubic inches, or ten pounds, or 70,000 grains, and the United States, or wine, gallon, as 231 cubic inches, or 58,372 grains, of pure water, at 4 degrees C.

Until quite recently, the standard method of reporting adopted by English and American chemists has been in grains per Imperial gallon; but at a late meeting of the Chemical Section of the American Association for the Advancement of Science, held in Buffalo, it was decided to report all water-analyses in parts per thousand, hundred-thousand and million, and circulars were sent to the different chemists throughout the country, asking their co-operation in establishing this uniform practice.

To chemists who have been using the old method, the transition will be simple, as the Imperial gallon of water weighs 70,000 grains; and by taking 70 cubic centimeters (70,000 milligrammes) of water for analysis, we have a miniature gallon, in which the milligramme corresponds to the grain.

Uniformity in reporting results of chemical analyses is of great advantage for comparison. It seems to some desirable to have a well-known volume in mind when looking over analyses. But "grains per gallon" does not ordinarily mean grains per U. S. standard, or wine, gallon, but grains per Imperial gallon (not a well-known volume to us), and moreover, parts per 100,000 can easily be converted into grains per gallon, if desired, by dividing the results by 10 and multiplying by 7. The practical usefulness in this instance, however, of the conception of the gallon as a unit-volume, is doubtful. The largest proportion of impurities found in any water used for potable or manufacturing purposes (except for making steam) is so small that the measure of volume is only comparative. For instance, 70 parts per 100,000, the greatest amount of impurities that could ordinarily be allowed in potable water, only means 7–100 of one per cent.; so that in practice the figures come to have significance, not in their relations to the unit-volume, but only by comparison with one another, or with certain standards.

Different methods of analysis give different results. This is especially true in the determination of the ammonias contained in water, where the varying amounts of water taken for distillation, and the varying proportions distilled over, yield widely different results; so that the results from each method of treatment and analysis pursued must be interpreted before comparison can be made with the results from other methods.

Sampling.—A sample of water taken for analysis should be at

least one-half gallon, if practicable. It should be kept in a cool and dark place until examined, and where possible, the analysis should be commenced within a few days. (In case of necessity, if extreme care be taken, as small a quantity as 50 cubic centimeters, or two fluid-ounces, may suffice for analysis.) It should be collected in a *perfectly clean* glass-stoppered bottle,* which should be rinsed out with the water before finally taking the sample. To insure a fair average for such a test, if the water be taken from a pump or faucet, it should be allowed to run some little time before collecting the sample. Samples should not be taken too near the shore of a pond or stream of water, if the average of the whole is desired.

In collecting, the mouth of the bottle should be opened some little depth under the surface of the water, to avoid the entrance of any surface-scum. A note should also be made, whether there has been previous long drought or heavy rain, and what is the condition of the water-level, together with a statement as to the neighborhood of any probable source of contamination.

Sanitary Analyses.—The difference in the specific gravity of waters, due to their relative degrees of purity, is very slight; so slight that, between pure distilled water and water that is very bad and has over 200 parts per 100,000 of total solid residue, the difference is in the fourth decimal place, and very close to the factor of error or variance in making specific-gravity tests. Hence no reliable tests can ordinarily be made in this way as to the comparative purity of samples of water.

The impurities in suspension vary from practically nothing in pure spring- and deep lake-waters, to perhaps 500 parts per 100,000 in the water of many rivers, after heavy freshets.

In solution, some pure artesian well-waters, some spring-water, rain-water, etc., have as little as 5 parts per 100,000, while the analyses of sea-water, some lake and river waters, and of some deep wells, range far above this proportion. Sea-water contains about 4100 parts.

For sanitary purposes, it is necessary to analyze water for the presence of metallic, and, occasionally, of vegetable, poisons, mineral acids, or septic poisons from contaminations of sewage. Such septic poisons cannot be separated, and their weight directly determined by analysis; for they never have a definite and exact chemical composition, like metallic poisons.

* Although it is preferable to have a glass-stopper to the bottle, still a clean and new cork will answer, but a cork contaminated with any other substance or luted with any sort of paste, might seriously injure the water for analysis.

The presence of septic poisons in water is shown by an excess of nitrogenous matter, indicated by the amount of chlorine and of free and albuminoid ammonia in the solution; the chlorine largely coming from the chlorides of sodium and potassium, which, with urea, are among the principal ingredients of ordinary urine.

Ammonia is evolved as the result of the decomposition of nitrogenous organic matter. Its presence in water in the small amounts reported in the analyses, even of the foulest waters, is harmless; it is only as indicative of decomposing organic matter that it shows the poisonous nature of the water in which it is found in considerable proportions.

Albuminoid ammonia is the term given to nitrogenous organic matter of such a character that it is converted into ammonia by letting the water-solution of it stand.

In large cities it is estimated that 30 gallons of water *per capita* of population, on the average, flows through the service-mains daily. A very large portion of this reaches the sewers and constitutes the bulk of the sewage. As a simple calculation, based upon the nature and amount of the contaminating material, will show, the average sewage contains in 100,000 parts, about 10 parts of free ammonia, and from 0.01 to 0.06 parts of albuminoid ammonia. When the free ammonia exceeds 0.02 parts per 100,000 in water taken from a thickly populated district, it is a probable sign that the water consists of dilute sewage. In such cases, the water will also usually contain over 0.6 parts of chlorine for river-water (or other water exposed to sunlight and atmospheric influence), and over 0.8 parts of chlorine for deep well- or spring-waters. The presence of albuminoid ammonia (over 0.018 parts per 100,000) with but little free ammonia and chlorine, is generally an indication of vegetable contamination, injurious to health. A sample of water containing per 100,000 more than 0.7 parts of chlorine, together with more than 0.015 parts of free and albuminoid ammonia, or both, can be declared with a considerable degree of certainty to contain decomposing nitrogenous organic matter which, either in itself or in the accompanying germs, contains septic poisons and will be dangerous to drink, although the character of the organic contamination is so varied that no exact danger-limit can, with our present knowledge of these impurities, be positively fixed. Not that the presence of chlorine, which is so often present in combination with sodium, as common salt, is dangerous in itself, or that minute amounts of free and albuminoid ammonia poison the water; but that they are indications of the presence of septic poisons, which cannot be separated by themselves.

"Nesslerizing" is a method discovered by the chemist Nessler, for determining the strength of dilute solutions of ammonia by comparing the yellowish-brown color given to the ammonia-solution by the Nessler reagent,—an alkaline solution of iodide of potassium saturated with periodide of mercury,—the color being more or less intense, according to the proportion of ammonia present in the solution tested. In testing samples of water, they are first distilled, and the ammonia which is carried over in the distillate is then determined by Nesslerizing. The ammonia found in the first distillate, before the addition of any reagents, is the free ammonia; the albuminoid is distilled over after the addition of caustic potash and permanganate of potash to oxidize the nitrogenous matter that is capable of producing ammonia on standing. It is called "albuminoid" ammonia, because albumen in water-solution, when treated as described above, yields ammonia.

The free ammonia represents the amount of change or complete decomposition of nitrogenous matter which has already taken place in the water; the albuminoid ammonia, the amount of change of which the water is still capable by reason of the presence of nitrogenous matter.

Complete oxidation of animal or other nitrogenous matter will result in the formation of nitric and carbonic acids, which, in the presence of alkalies or alkaline earths, will form nitrates and carbonates, comparatively harmless in themselves.

In some waters, where the oxidation of the impurities has been considerable, the presence of nitrates, in the proportion of over 0.40 parts per hundred thousand in pond- or river-waters and of over 3.00 parts for well-waters, is almost the only indication shown by the chemical analysis of the known contamination of the water with decomposing nitrogenous matter.

Water that has percolated through a loamy soil rich in decomposing organic matter will be found generally to be high in contained nitrates.

The presence of nitrites and of ammonia shows intermediate states of decomposition of the organic matter, due, very likely, to the microbes and living germs of disease, which are nearly always present with decomposing organic matter. Nitrous acid, which combines with alkalies and alkaline earths to produce nitrites, is an intermediate state between ammonia and nitric acid salts or nitrates.

The metallic poisons apt to be found in potable waters are lead, zinc, iron, and copper. Chromium and arsenic are rarely found, and their presence in water, in any quantity, should be considered dangerous.

Authorities differ as to the amounts of metal which would be considered injurious. Wanklyn states that "a good drinking-water should not contain more than one-tenth or two-tenths grains of iron per gallon, or more than one-tenth of lead or copper." It is a question whether so small a quantity of iron should condemn a water. Many iron-springs, considered healthful, contain a much larger amount. Undoubtedly, the form in which the iron-salt exists would largely influence its action.

Well-authenticated cases of lead-poisoning have been occasioned by the cumulative action of lead which had been taken into the system in water containing 0.2 parts of lead per 100,000.

A very simple test for metals can be made by taking a miniature gallon of the water (70 cubic centimeters) in a clean porcelain dish, adding three drops of sulphide of ammonium, which should give but a very slight coloration to the fluid, and which should be instantly cleared up and become invisible by the addition of a few drops of hydrochloric acid. Good water will answer to this test, and water that does not should be examined carefully and quantitatively for the presence of metallic impurities, before being accepted as potable.

Microscopic Examination of Water.—There are many living organisms, both animal and vegetable, which infest some waters, and which have to be examined microscopically. The microbes, ferments, and germs existing in some waters are the causes of many diseases flesh is heir to. Every year scientific research, especially with the aid of the microscope, is bringing to light new facts as to the presence and influence on human life and health of these minute organisms.

The microscopic examination of water is not so much to determine the larger forms of animal and vegetable life, as it is to separate and identify the septic or pathogenic microbes that are due to sewage-contamination, or are taken up from the air and soil in fever- and cholera-infected districts. Decomposing nitrogenous organic matter we cannot say positively is poisonous, but we know by the aid of the microscope that associated with it are many microbes, some of which may be germs of specific diseases.

These microbes are the different forms of *Bacillus*, as the *B. tuberculosis*, which is the organism causing tubercular consumption; the "*Comma Bacillus*" of Koch, which is now generally conceded to be the cause of Asiatic cholera, and others which have not been so positively identified, such as the *Bacilli* of typhoid and relapsing fevers, and that of yellow fever, which Dr. Freire, of Brazil, claims to have identified and cultivated. These organisms are only found in waters polluted with surface drainage and sewage contaminated

by the *dejecta* of fever patients, the *sputum* of persons suffering from tubercular consumption, or similar matter expelled from the bodies of patients in some of the many diseases caused by the presence of pathogenic microbes. These dangers are now so universally recognized, that State Boards of Health are publishing very stringent rules for the destruction of such poisonous matter, and, indeed, of anything with which it may have come in contact.

The investigation and the determination of the different forms of microbes require the highest skill of the trained microscopist; and any statements made by investigators without such training and skill are very likely to be erroneous. These organisms are among the smallest and least differentiated with which the microscopist has to deal. They are distinguished by different methods of cultivation, and by the way in which the different groups take certain well-known and distinctive stains; the water-soluble aniline dyes being those generally used for this purpose.

Micro-organisms are classified as aerobic and anaerobic (or those which require oxygen for their development and those which do not), septic and non-septic, pathogenic and non-pathogenic; the septic being those which cause putrefaction in dead bodies, while the pathogenic are those which cause disease.

These minute organisms may be separated from water by adding about one cubic centimeter of a one and one-half per cent. solution of osmic acid to 30 or 40 cubic centimeters of water. All organisms are by this reagent precipitated as a thick, dark-colored deposit, from which the water is decanted off, leaving the deposit to be examined afterwards under the microscope. In this deposit will be found a large number of septic and non-septic organisms which occur in all surface waters; and without careful training in a biological laboratory, it will be impossible to distinguish the various forms.

Cultivation-experiments are carried on, either by plate- or "flask-" culture, in the different organic or inorganic media which have been proved suitable for the purpose. A list of such media can be found in any of the standard works on microscopy; most fully, perhaps, in Klein's "Micro-Organisms and Disease," fourth edition. The methods of sterilizing all the apparatus and media, and of preserving them from contamination by the air, or from any other cause, are there explained.

After inoculating the culture-medium, the growth of the colonies is carefully watched for several days; and if the colonies appear to be pure—that is, composed of only one microbe—fresh tubes or

plates are inoculated, and this is repeated several times, until there is no doubt of the purity of the culture.

The microbes must then be studied by their reaction with certain stains, and by their effect upon living animals. The number of organisms in water is estimated by inoculating a certain amount of sterilized gelatine on a plate with a measured amount of water and counting the number of colonies which are developed.

The Total Solid Residue.—The total solid residue in water varies from 5 parts per 100,000 in some deep, pure spring-waters to 4100 parts in ocean-water. The character of this residue, as well as its amount, should be known, to establish the value of water for drinking. In most cases, where the total solid residue left upon evaporation does not exceed 40 parts per 100,000, it is not found upon examination to injure the water for such purposes. A larger amount than 40 parts per 100,000 should occasion a further investigation of the water. Of course, if the total solid residue consists of very poisonous matter, a smaller amount will cause injury.

The solid residue may consist of lime, soda, magnesia, potash, fine argillaceous matter held in suspension, or organic matter, and of finely divided silica. The chloride, bicarbonate, and sulphate are the salts of lime most commonly looked for in waters to be tested as to fitness for drinking and cooking. The presence of more than 15 parts per 100,000 of these lime salts should condemn a water for such purposes, under ordinary circumstances. However, in a good many of the Western states, the people are obliged to use water containing a larger amount of lime than this. A large amount of lime in drinking-water is liable to produce *goitre*, calculus, urinary and dyspeptic diseases. On the other hand, the presence of lime salts in sufficient quantity to render the water perceptibly hard, yet less than 35 parts per 100,000, has not proved unwholesome. In fact, many cities having hard drinking-water have a very favorable mortality as compared with similar cities having soft drinking-water, carrying less than 15 parts per 100,000 of lime salts.

The following definition of liquids which should be deemed polluting and inadmissible into a stream, was formulated by the Rivers Pollution Commission of Great Britain (1886).

(a.) Any liquid which has not been subjected to perfect quiet in subsidence-ponds of sufficient size for a period of at least six hours, or which, having been so subjected to subsidence, contains in suspension more than one part by weight of dry organic matter in 100,000 parts by weight of the liquid, or which, not having been so subjected to subsidence, contains in suspension more than three parts

by weight of dry mineral matter, or one part by weight of dry organic matter in 100,000 parts by weight of the liquid.

(b.) Any liquid containing, in solution, more than two parts by weight of organic carbon or 0.3 part by weight of organic nitrogen in 100,000 parts by weight.

(c.) Any liquid which shall exhibit by daylight a distinct color when a stratum of it one inch deep is placed in a white porcelain or earthenware vessel.

(d.) Any liquid which contains in solution, in 100,000 parts, by weight, more than two parts by weight of any metal except calcium, magnesium, potassium, or sodium.

(e.) Any liquid which in 100,000 parts by weight contains, whether in solution or suspension, in chemical combination or otherwise, more than 0.05 part by weight of metallic arsenic.

(f.) Any liquid which, after acidification with sulphuric acid contains, in 100,000 parts by weight, more than one part by weight of free chlorine.

(g.) Any liquid which contains, in 100,000 parts by weight, more than one part by weight of sulphur, in the condition of either sulphuretted hydrogen or a soluble sulphuret.

(h.) Any liquid possessing an acidity greater than that which is produced by adding two parts by weight of real muriatic acid to 1000 parts by weight of distilled water.

(i.) Any liquid possessing an alkalinity greater than that which is produced by adding one part by weight of dry caustic soda to 1000 parts by weight of distilled water.

(k.) Any liquid exhibiting a film of petroleum or hydro-carbon upon its surface, or containing in suspension, in 100,000 parts, more than 0.05 part of such oil.

To these standards was attached the proviso, that "no effluent water shall be deemed polluting if it be not more contaminated with any of the above named polluting ingredients than the stream or river into which it is discharged."

Mineral Waters.—Natural spring-waters impregnated with mineral substances to such an extent as to have a medicinal effect, are called mineral waters. Such waters are commonly divided, according to the character of the foreign substances held in solution, under the heads of carbonated, alkaline, sulphuretted, saline (including magnesian, chalybeate, and chlorinated), and siliceous waters. It is not necessary to describe these various classes here. The subject is one, with the general features of which our members are presumed to be acquainted, and the thorough discussion of which, though it

would be both pertinent and valuable, is not within the purpose of the present paper.

Commercial Analyses.—By far the most common commercial analysis of water is made to determine its fitness for making steam. Water containing more than 5 parts per 100,000 of free sulphuric or nitric acid is liable to cause serious corrosion, not only of the metal of the boiler itself, but of the pipes, cylinders, pistons and valves with which the steam comes in contact. Sulphuric acid is the only one of these acids liable to be present in the water from natural sources; it being often produced in the waters of the coal and iron districts, by the oxidation of iron pyrites to sulphate of iron, which, being soluble, is lixiviated from the earth strata and carried into the stream; the presence of organic matter, taken up by the water in its after-course, reducing the iron and lining the bottom of the stream with red oxide of iron, and leaving a considerable proportion of the sulphuric acid free in the water. This is a troublesome feature with the water necessarily used in many of the iron districts of this country. The sulphuric acid may come from other natural chemical reactions than the one described above. Muriatic and nitric acids, and often sulphuric acid as well, may be conveyed into water through the refuse of various kinds of manufacturing establishments discharged into it.

The large total residue in water used for making steam causes the interior linings of the boilers to become coated, clogs their action and often produces a dangerous hard scale, which prevents the cooling action of the water from protecting the metal against burning.

Lime and magnesia bicarbonates in water lose their excess of carbonic acid on boiling, and often, especially when the water contains sulphuric acid, produce, with the other solid residues constantly being formed by the evaporation, a very hard and insoluble scale. A larger amount than 60 parts per 100,000 of total solid residue will ordinarily cause troublesome scale, and should condemn the water for use in steam-boilers, unless a better supply cannot be obtained.

The following is a tabulated form of the causes of trouble with water for steam-purposes, and the proposed remedies, taken from notes of Prof. W. R. Nichols's lectures on "Industrial Chemistry:"

Brief Statement of Causes of Incrustation.

1. Deposition of suspended matter.
2. Deposition of dissolved salts from concentration.
3. Deposition of carbonates of lime and magnesia by boiling off carbonic acid, which holds them in solution.

4. Deposition of sulphates of lime, because sulphate of lime is but slightly soluble in cold water, less soluble in hot water, insoluble above 140 degrees C.

5. Deposition of magnesia, because magnesium salts decompose at high temperature.

6. Deposition of lime soap, iron soap, etc., formed by saponification of grease.

Various Means for Preventing Incrustation.

1. Filtration.
2. Blowing off.
3. Use of internal collecting apparatus or devices for directing the circulation.
4. Heating feed-water.
5. Chemical or other treatment of water in boiler.
6. Introduction of zinc into boiler.
7. Chemical treatment of water outside of boiler.

Tabular View.

Troublesome Substance.	Trouble.	Remedy or Palliation.
Sediment, mud, clay, etc.	Incrustation.	Filtration. Blowing off.
Readily soluble salts.	"	Blowing off.
Bicarbonates of lime, magnesia, iron.	"	Heating feed. Addition of caustic soda, lime, or magnesia, etc.
Sulphate of lime.	"	Addition of carbonate of soda, barium chloride, etc.
Chloride of magnesium and sulphate of magnesia.	Corrosion.	Addition of carbonate of soda, etc.
Carbonate of soda in large amounts.	Priming.	Addition of barium chloride, etc.
Acid (in mine waters).	Corrosion.	Alkali.
Dissolved carbonic acid and oxygen.	"	Heating feed. Addition of caustic soda, slacked lime, etc.
Grease (from condensed water).	"	Slacked lime and filtering. Carbonate of soda. Substitute mineral oil.
Organic matter (sewage).	Priming.	Precipitate with alum or ferric chloride and filter.
Organic matter.	Corrosion.	Ditto.

The mineral matters causing the most troublesome boiler-scales are bicarbonates and sulphates of lime and magnesia, oxides of iron, alumina, and silica. The analyses of some of the most common and troublesome boiler-scales are given in the following table:

Analyses of Boiler-Scale.

	Sulphate of Lime.	Magnesia.	Silica.	Peroxide of Iron.	Water.	Chloride of Sodium.	Carbonate of Lime.	Lime, CaO.	Sulphuric Acid.	Carbonic Acid.	Authority.
From British Admiralty Report. Steamer											
Amethyst.....	94.64	2.88	trace	trace	2.45	trace					Admir'lty Report.
Malabar.....	95.93	3.10	0.40	0.40	.....	"					"
Fox.....	77.30	10.55	trace	5.80	6.00	"					"
Patrothus.....	96.37	1.62	0.60	0.60	1.-5	"					"
Velindra.....	91.38	2.31	1.40	1.60	3.30	"					"
Other Sea-Going Scale. (Wallace.) Atmospheric Pressure of											
0.34	33.95	40.05	trace	1.33	24.67	trace	.....				Wallace.
0.4 to 0.51	66.88	18.96	"	0.50	11.66	"	0				"
0.7	69.77	15.75	0.16	1.36	8.25	0 99	3.44				"
1 to 1.12	74.21	14.95	0 57	1 34	6.89	2.04	.....				"
1.4	72.85	13.18	0.80	2.40	8.27	2.16	0.34				"
Hartford, Conn., River Water.....	22.70	5.32		4.57				40.0	2.63	15.62	Mac Farlane.
Same as above after using Soda Ash....	18.55	3.33		2.23				42.35	.....	24.95	"
Holyoke, Mass.....	12.49	7.34		3.57				29.41	0.64	25.03	"
Southington, Conn., Well Water I.....	23.50	10.66		0.60				34.83	6.89	18.74	"
II.....	28.85	5.58		1.01				40.90	2.33	17.03	"
Danbury, Conn., Brook	15.07	4.94		12.77				34.66	0.48	21.15	"
North Adams, Mass., Brook	4.23	4.27		1.12				35.01	46.51	.....	"
St. Croix River.....	0.61	4.80		1.12				35.49	52.03	.....	"
Atmospheric Pressure.											
2.5 to 3	50.75	1.19	0.47	2 24	3.68		44.25				Fischer.
3	67.14	9.69	trace	5.28	7.90		14 65				"
3.5	79.71	3.83	0.88	1.67	3.45		8.30				"
3.5	31.96	7.11	trace	2.10	2.31		55.65				"
5	80.74	4.36	.....	0.52	4.21		8.20				"
5—6	98.04	.....	.....	0.20	.....	.....	.....				"
N. Y. C. & H. Ry., No. 1...	74.07	9.19	0.65	0.08	1.14		14.78				Chandler.
" " 2...	71.37		1.76								"
" " 3...	62.86	18.95	2.60	0.92	1.28		12.62				"
" " 4...	53.05		4.79								"
" " 5...	46.83		5.32								"
" " 6...	30.80	31.17	7.75	1.08	2.44		26.93				"
" " 7...	4.95	2.61	2.07	1.03	0.63		86.25				"
" " 8...	0.88	2.84	0.65	0.36	0.15		93.19				"
" " 9...	4.81		2.92								"
" " 10...	30.07		8.24								"

The following table shows the analyses of samples of water giving bad results for steam-purposes:

Analyses in Parts per Hundred Thousand of Waters Giving Bad Results for Steam-Purposes.

	Bicarbonate of Lime Deposited on Boiling.	Bicarbonate of Magnesia Deposited on Boiling	Total Lime.	Total Magnesia.	Sulphuric Acid.	Chlorine.	Iron.	Organic Matter.	Alumina.	Chloride of Sodium.	
Feed-water giving											
Scales 2.5—3...	225	19	450	85	219	293					Fischer.
" 3 atmosph.	88	3	147	22	121	59					"
" 3.5 "	tr'ce	0	46	9	40	.....	See Table of Boiler Scales.				"
" 3.5 "	63	39	155	68	89	91					"
" 5 "	46	0	244	32	232	9					"
" 5—6 "	tr'ce	0	599	81	306	770					"
Coal-mine wat'r	110	25	119	39	890	590	780	30	640		A. E. Hunt.
Salt-well.....	151	38	1.90	48	360	990	38	21	30	13.10	A. E. Hunt.
Spring.....	75	89	95	120	310	21	75	10	80	36	"
Monongahela River*.....	130	21	161	33	210	38	70				"
"	80	70	94	81	219	210	90				"
"	32	82	61	1.04	28	1.90	38				"
Allegheny riv'r near Oil-w'rks	30	50	41	68	890	42	23				"

Many substances have been added with the idea of causing chemical action which will prevent boiler-scale. As a general rule, these do more harm than good; for a boiler is among the worst possible places in which to carry on a chemical reaction, where it nearly always causes more or less corrosion of the metal, and is liable to cause dangerous explosions.

In cases where water containing large amounts of total solid residue is necessarily used, a heavy petroleum oil, free from tar or wax, which is not acted upon by acids or alkalies, not having sufficient wax in it to cause saponification, and which has a vaporizing-point at nearly 600 degrees Fahrenheit, will give the best results in preventing boiler-scale. Its action is to form a thin greasy film over the boiler-linings, protecting them largely from the action of acids in the water, and greasing the sediment which is formed, thus preventing the formation of scale and keeping the solid residue from

* Taken near discharge-pipes from large manufacturing establishments.

the evaporation of the water in such a plastic, suspended condition that it can be easily ejected from the boiler by the process of "blowing off." If the water is not blown off sufficiently often, this sediment forms into a kind of a "putty" that will necessitate cleaning the boilers. Any boiler using bad water should be blown off every twelve hours. The formation of scale of hard residue on the boiler-linings will be almost entirely prevented by the use of this oil, which is manufactured in Pittsburgh, and sold in the markets under the name of "Boiler Scale Resolvent."

Hardness of Water.—The hardness of water, or its opposite quality, indicated by the ease with which it will form a lather with soap, depends almost altogether upon the presence of compounds of lime and magnesia. Almost all soaps consist, chemically, of oleate, stearate, and palmitate, of an alkaline base, usually soda and potash. The more lime and magnesia in a sample of water, the more soap a given volume of the water will decompose, so as to give insoluble oleate, palmitate, and stearate of lime and magnesia, and consequently the more soap must be added to a gallon of water in order that the necessary quantity of soap may remain in solution to form the lather. The relative hardness of samples of water is still generally expressed in terms of the number of standard soap-measures consumed by a gallon of water in yielding a permanent lather, rather than in parts per hundred thousand, for the reason that as the relative hardness of samples of water has come to be a commercial quality, and is determined by many who are not chemists, it has been found more difficult to change the method of reporting. As before explained, reports in degrees or grains per imperial gallon can be readily transposed in parts per hundred thousand by multiplying the number of grains by 10 and dividing the product by 7.

The standard soap-measure is the quantity required to precipitate one grain of carbonate of lime.

It is commonly reckoned that one gallon of pure distilled water takes one soap-measure to produce a lather. Therefore, one is deducted from the total number of soap-measures found to be necessary to use to produce a lather in a gallon of water, in reporting the number of soap-measures, or "degrees" of hardness of the water sample. In actually making tests for hardness, the "miniature gallon," or seventy cubic centimeters, spoken of on a previous page, is used rather than the inconvenient larger amount. The standard measure is made by completely dissolving ten grammes of pure castile soap (containing 60 per cent. oleic acid) in a liter of weak alcohol (of about

35 per cent. alcohol). This yields a solution containing exactly sufficient soap in one cubic centimeter of the solution to precipitate one milligramme of carbonate of lime, or, in other words, the standard soap-solution is reduced to terms of the "miniature gallon" of water taken. The correctness of the standard soap-solution is standardized against a solution of one gramme of carbonate of lime, dissolved in hydrochloric acid, carefully neutralized with ammonia and diluted to one liter, so that each cubic centimeter represents one milligramme of carbonate of lime.

The standard soap solution should be standardized against a known lime solution of a strength approximating that of the samples of water to be analyzed; especially should this be done with waters strong in contained lime salts.

If water charged with bicarbonate of lime, magnesia, or iron, is boiled, it will, on the excess of the carbonic acid being expelled, deposit a considerable quantity of the lime, magnesia, or iron, and consequently the water will be softer. The hardness of the water after this deposit of lime, after long boiling, is called the *permanent hardness*, and the difference between it and the total hardness is called *temporary hardness*.

Lime salts in water react *immediately* on soap-solutions, precipitating the oleate, palmitate, or stearate of lime at once. Magnesia salts, on the contrary, require some considerable time for reaction. They are, however, more powerful hardeners; one equivalent of magnesia salts consuming as much soap as one and one-half equivalents of lime.

The presence of soda and potash salts softens rather than hardens water.

The applicability of a given water for very many of the manufacturing industries requires its analysis for the contained impurities. The most common injurious impurities for manufacturing purposes are the free acids, excessive amounts of total solids, and salts affecting the relative hardness.

ASPHALT AND ITS USES.

BY F. V. GREENE, NEW YORK CITY.

(Buffalo Meeting, October, 1888.)

THIS paper is based on my experience in the use of asphalt, for paving and other purposes, during the last ten years, part of the

time as Government engineer, in charge of such work in Washington, and part of the time as the managing officer of a corporation engaged in laying such pavements in a large number of cities. It is presented in obedience to the invitation of the Council of the Institute, which must, therefore, assume the responsibility of deciding that the subject will prove appropriate and interesting.

Of all the materials used in construction, I think it may fairly be said that the one more nearly indispensable than any other is cement. The sticking- or binding-material, which causes the larger masses of sand, stone, or brick to adhere and hold firmly together, is, and always has been, so universally employed, that we can hardly conceive of any permanent structures erected without it.

There are two general classes of cements, viz., hydraulic and bituminous. In the one, water is employed to make a paste which, in drying, undergoes certain chemical changes and becomes a hard and adhesive artificial stone. Owing to the fact that it is not materially affected by water or heat after it has become hard or "set," it is, in some respects, superior to bituminous cement, and is much more widely used. In the other, the hardening and adhesive qualities are obtained by simply cooling from a liquid to a solid condition. This cement is again affected by heat, and hence is not adapted to constructions which are to be subjected to wide ranges of temperature. On the other hand, it has the advantages of being water-proof and of being slightly ductile. Hence its use is restricted to those structures in which rigidity is undesirable, which are subjected to heavy blows and concussion, and in which complete impermeability to water is necessary. Within this restricted class of structures its superiority as a cementing material is uncontested.

It is with the bituminous cements, and with asphalt as one of that class, that we are now concerned.

While the hydraulic cements have been carefully studied by Vicat, Paisley, Gillmore, and others, who have thoroughly classified them and given them a distinct and well understood nomenclature, the bituminous cements, owing to their restricted use, have not been so well defined, and a certain amount of confusion exists as to the meaning of the words, asphalt, tar, bitumen and pitch.

Mr. Leon Malo, the well-known engineer of the Seyssel mines of France, ignoring the Greek lexicon, and the meaning which has attached to the word for over twenty centuries, has sought to appropriate for the mines of his company the exclusive ownership of the word "asphalte," and claims that this word describes only a bitu-

minous limestone. On the other hand, the various tar-poultices, made from the refuse of the nearest gas-works, which were introduced into Washington, in 1871, and spread thence over the country like an epidemic, more terrible than a pestilence to the taxpayer, were all known as "asphalt pavements." A reference to Worcester's, or any other standard dictionary, or to the standard works on mineralogy, will correct these errors. *Asphaltos* is an ancient Greek word which passed into the Latin as *asphaltum*. In the Greek it is used by Homer; in the Latin version of the Bible it appears in the book of Genesis in describing the construction of the ark and the tower of Babel; in English, it is employed by Milton in *Paradise Lost*, and everywhere it is used to describe just what Worcester defines it to be, viz., native bitumen or mineral pitch. He derives it from *a* privative and *σφάλλειν*, to slip, in view of its ancient use as a cement; but it may not have been originally a Greek word at all.

It is found in various parts of the world, either combined with other substances or in a free state, in various degrees of consistency from complete fluidity to a substance resembling in appearance hard rubber or black sealing-wax.

Of its use by the ancients we know little beyond the tradition that the walls of Babylon were cemented with asphalt from a neighboring spring on the Euphrates. It is also reported that it was used by the Egyptians in embalming; but the exact manner in which they profited by its antiseptic qualities is not now known.

In order to arrive at any classification it is necessary to give some definition to the word bitumen, which is not strictly defined in any of the books. Possibly, as good a definition as can be given is to say that bitumen is the generic name for the entire class of hydrocarbons remaining from the distillation of coal, or other substances from which coal has been formed. The three words, oil, tar, and pitch, are used to distinguish the three varieties of free bitumen which are liquid, viscous, and solid. The *native mineral oil*, or liquid bitumen, has received the name of petroleum; *native mineral tar*, or viscous bitumen, has only been found in such limited quantities and has been so little used that it has received no specific name; *native mineral pitch*, or hard bitumen, *i.e.*, bitumen found in nature in the solid form, has always been known as asphalt, or asphaltum.

With these fundamental definitions, it is not difficult to construct a tabular statement showing the varieties of bitumen and its compounds, and the localities where they are found, as follows. This

table, though similar in form, differs in arrangement from the table given by Malo:

- Bitumen:
  - 1. In a free state:
    - Liquid: Petroleum (Pennsylvania, Ohio, etc., Baku, on the Caspian).
    - Viscous:
      - Tar-Springs (France, Alabama, Venezuela).
      - Gas Tar.
    - Solid: Glance Pitch (Cuba, Texas, Utah).
  - 2. Mixed with earthy matter: Asphalt (Trinidad Pitch Lake).
  - 3. Combined with silica: Bituminous Sandstones (France, California, Utah).
  - 4. Combined with lime: Bituminous Limestones (France, Germany, Spain, Sicily, Utah, Colorado, Cuba, Mexico).

Petroleum.—This substance contains so many volatile oils that it is liquid at ordinary temperatures, and hence cannot perform any of the functions of a cement. No discussion of it is therefore pertinent in this paper. It is proper to remark, however, that its residuum, after extracting the lighter oils, is used to temper asphalt in order to obtain the proper degree of ductility; and if this residuum is still further distilled, a pitch can be obtained which has all the properties of native mineral pitch or asphalt, and is also chemically pure, *i.e.*, wholly soluble in carbon disulphide. The exact nature of this final distillation, or process of treatment, has never been made public. It is so expensive as to be wholly prohibitory for paving- and roofing-compounds, but the resulting product gives, I believe, the highest tests yet obtained as an insulator for underground wires. It is used in the Waring cable.

Tar-Springs.—These "springs," which give forth a native mineral tar of about the consistency of thick molasses, are found near the headwaters of the Loire, in France, and in various parts of Alabama, Tennessee, and Kentucky. The material exudes from between thin layers of limestone, and is quite pure, about 91.9 per cent. being soluble in carbon disulphide. The quantity, however, is extremely limited, each spring yielding only a few barrels in a season; and it is not commercially practicable to mine the rock for the purpose of extracting the small amount of tar scattered through it.

The springs of Venezuela are larger in extent, but are remote of access, and uprofitable for commercial purposes on account of the expense of shipping. They have never been used to any large extent, and hence their practical value is undetermined. The bitumen is quite pure, but contains a considerable amount of water, which makes it dangerous to handle, since in the process of refining it

"foams" (*i.e*, increases in volume owing to escaping vapor) to such an extent as to overflow the kettles and take fire.

Gas-Tar.—This is the well-known product resulting from the distillation of bituminous coal in the manufacture of illuminating-gas. Twenty-five years ago, it was considered as refuse and the gas companies paid for hauling it away. Since then, the chemistry of the hydrocarbons has made such progress that it is now commonly reported that the by-products made from the tar pay all the expenses of gas-making and leave the gas as pure profit. These various products, creosote, fuel-oil, naphtha, anthracene, aniline, etc., are well known. The only ones that have any cementing qualities are the residuum of distillation at various points, known in commerce as roofing- and paving-pitch. These resemble the native pitch (*i.e.*, asphalt) so closely in outward appearance that they were thought at one time to be equally valuable; but experience has shown that this is not the case. These products are the result of a destructive distillation, arbitrarily stopped at a certain point. Their chemical arrangement has been disturbed and has not reached a stable compound. Under the influence of heat and moisture, when exposed to the air, more oxygen is absorbed and a slow distillation takes place, which in time becomes an ultimate distillation, *i.e.*, reduces the substance to a friable coke, deprived of all cementing qualities.

In situations where these substances are not subjected to the influences of the air, the cementing quality is retained for many years; but the presence of an unlimited supply of oxygen and the sun's heat causes them to deteriorate rapidly. To overcome this tendency, the practice has arisen of mixing the gas-tars with genuine asphalt; and this has been successful in proportion to the amount of asphalt used—most successful of all when the asphalt is 100 per cent. and the tar 0. The use of gas-tar as a cement in roofing- and paving-compounds is governed by substantially the same principles as the use of asphalt for these purposes, which will be hereafter described.

Glance Pitch.—The hard asphalt, commonly known as *Glance Pitch*, is extremely pure, 98.9 per cent. being soluble in CS_2. This substance is very fragile and brittle, crumbling easily between the fingers, and is therefore useless for paving- or roofing-purposes; but its purity and its stable character in a chemical sense make it very valuable for varnish. It is prepared for this purpose by dissolving it in spirits of turpentine. It is found in Texas and in various parts of Mexico and Cuba, and also in Utah, where it has been known under the name of Gilsonite.

The substances named above comprise all the principal known varieties of bitumen in a free state. The asphalt of Trinidad, as will be subsequently explained, is mixed with a certain amount of earthy matter. The combinations of bitumen with quartz or limestone, forming the bituminous rocks, have had a wide development in commercial use, and will now be described.

Bituminous Sandstones.—The bituminous sands and sandstones of France yield, on analysis, from 5 to 13 per cent. of bitumen, mixed with sand and a small amount of limestone and clay. They have been principally used for the purpose of extracting the bitumen. The process consists in boiling with water at a temperature which fuses the bitumen and allows the sand to settle to the bottom; the water is then evaporated from the scum or froth, and the bitumen is left as a deposit. The greater portion of this supply has now been exhausted.

The bituminous sandstone of California is found in large quantities at various points between San Francisco and Los Angeles. It contains about 12 to 18 per cent. of bitumen, and the rest is quartz sand, in grains about $\frac{1}{10}$th of an inch in size. The material is sufficiently soft to yield to the heat and pressure of the hand. Within the last few years it has come into use for paving-purposes in Los Angeles and other cities on the Pacific coast. The rock is quarried, broken to fragments of about 2 inches in size, heated in kettles by steam (which causes it to fall into powder), and then, while still hot, taken to the street and compressed by rolling or tamping. None of the pavement has been laid on a large scale for longer than two years, and the reports as to its quality as a paving-material are conflicting. It appears to be so soft that wheels and horses' feet sink into it quite deeply; but, on the other hand, these marks appear to be more or less obliterated by the next passing vehicle.

Bituminous Limestones.—The bituminous limestones of France have furnished the material with which portions of the streets of Paris and other European cities are paved. The best mines are those of Seyssel and Val-de-Travers. They are of uniform quality, and yield on analysis about 10 to 11 per cent. of bitumen, the rest being pure carbonate of lime, free from clay or other impurities. The extent to which the material has been used for paving roadways is about as follows: In Paris, 13 miles; in London, 15 miles; in Berlin, 9 miles; in other cities, 10 miles; total, 47 miles.

It was introduced in Paris about thirty years ago, and has proved a durable material, with the qualities pertaining to a smooth pave-

ment. The chief objection to it is its slipperiness. The limestone, which forms nearly 90 per cent. of its volume, is not granular, but amorphous, and it polishes under traffic, so as to become at times dangerous for horses.

The preparation of the pavement consists in mining the rock and breaking it into pieces about 3 inches in size; these are passed through a set of toothed rollers, and then through smooth rollers, which reduce the material to a fine powder. This is then placed in a large iron cylinder into which hot air (about 500° F.) is introduced. The interior of the cylinder is provided with a large revolving helix or screw, of nearly the same diameter as the cylinder, and having several small paddles at its outer edges. These paddles raise the powder as the helix revolves, and the powder then falls through the hot air, and at the same time is moved by the helix towards the end of the cylinder. The material is fed at one end of the cylinder in certain proportions for each turn of the helix, and when the entire charge has been moved to the opposite end it falls into a box on the under side of the cylinder, from which it is removed, placed in carts, hauled to the street, spread on a bed of concrete previously prepared, and thoroughly compacted by hard tamping with iron rammers. It cools to the temperature of the atmosphere in a few hours, and the street is then thrown open to traffic.

The sidewalks of Paris are also paved with a preparation known as asphalt mastic, which is made in the following manner. The rock, after being reduced to powder as above described, is placed in cylindrical kettles, in which about 8 per cent. of Trinidad asphalt has previously been placed and melted. The mass is stirred by revolving arms and agitators, at a temperature of about 280° F., for about five hours. It is thus thoroughly "cooked," and is then run out of the kettles into moulds, where it cools in the form of cakes or blocks, weighing about 50 pounds each. These are sold in commerce. To use them on the street the material is again heated in a suitable kettle in the following proportions:

Mastic cakes (broken to egg-size), ...	60 pounds.
Trinidad asphalt, ...	4 "
Fine gravel and sand, ...	36 "
	100 "

This is "cooked" for about two hours at a temperature of about 300° F., great care being taken constantly to stir the mixture. It

is then taken out of the kettle by the bucketful and poured on the foundation previously prepared for the side-walk, its consistency being such that it will flow very slowly. It is then spread by means of wooden trowels, and compressed and smoothed by rubbing, as in plastering.

These side-walks have given great satisfaction in Paris, and are almost universally used throughout the city. Their superficial area is nearly 5,000,000 square yards (about fifteen times as great as the area of the asphalted streets), and their length probably exceeds 1000 miles.

The bituminous limestones of Germany, Sicily and Spain are of the same general character as those of France, but they are inferior in quality, since they are not so homogeneous or so uniformly impregnated with bitumen. Samples of a similar kind of limestone have been found in various parts of the Rocky Mountains, but they have not as yet come into general use for paving-purposes, and it is probable that the expense of transportation will prevent their introduction to any great extent.

The bituminous limestones of Mexico and Cuba contain about 70 per cent. of bitumen, mixed with limestone, sand, and other impurities. They have not come into general use for paving. It is found that, on partial reduction, the bitumen from this and other sources can be resolved into two substances, to which the names petrolene and asphaltene have been given, the petrolene being soft and resembling petroleum, and the asphaltene being hard and resembling asphalt.

In the Cuban rock the asphaltene is too largely in excess to make a good cementing-material. A pavement laid with Cuban asphalt in Washington, under guaranty of maintenance, was taken up by the contractor at the end of a year and relaid with Trinidad asphalt. In Louisville a pavement of Cuban asphalt was laid in 1882. Between one-third and one-half of it has been taken up and relaid with Trinidad asphalt, and what remains is everywhere cracked and shattered like a broken pane of glass. This material is evidently too brittle for a good paving-cement. It is also so situated geologically that its mining is very expensive.

Trinidad Asphalt.—The asphalt of Trinidad is found in a so-called "lake," situated about 100 feet above the sea, and about 3 miles from the shore of the island, at the village of La Brea (the Spanish word for pitch). Its area is about 114 acres; its depth, as far as ascertained by certain rude borings, is reported to be about 18 feet at the sides and 78 feet in the center; and underlying it there is

said to be a bed of blue clay. If these figures are correct, the lake contains about 6,000,000 tons of asphalt. Whether these borings are even approximately accurate is, however, very doubtful. It is even contended by some that the lake is still fed from underground sources. The only positive information on the subject is the fact that the excavations of the last ten years, amounting to about 180,000 tons, have not appreciably lowered its level; and for all practical purposes, at the present rate of consumption, it may be said to be inexhaustible.

The word "lake," applied to this deposit, is an entire misnomer. It is a level tract of brownish material having an earthy appearance. Cracks or fissures, having a width and depth of a few feet, appear here and there over the surface. Some of them are filled with rain-water, while others have been filled with soil blown there by the wind, and giving support to a scrubby vegetation. Some travellers have reported that the deposit is liquid in the middle, but such is not the fact. Carts and mules can be driven everywhere on its surface.

The material is dug with a pick and shovel, loaded into carts, and hauled to the beach. Here it is placed in baskets, which are carried by coolies wading through the surf to lighters, and from these lighters it is loaded on vessels. During the voyage the material unites into a solid mass, and has to be removed again by the use of the pick and shovel. On being unloaded it is placed for about five days in large tanks heated by a slow fire. The moisture is expelled, the roots of trees and other vegetable matters are skimmed off the surface, the earthy matter with which it is combined settles by gravity, and the refined product is run off into barrels. This refining is in reality a mere heating to a liquid condition, in order to allow the sediment to deposit; and great care is taken not to heat the material to a point which will in any way change its chemical condition, or produce distillation.

The crude asphalt has the following properties: Specific gravity, 1.28; hardness at 70° F., 2.5 to 3 in Dana's scale; color, chocolate brown. It yields on partial analysis:*

Bitumen,	39.83 per cent.
Earthy matter,	33.99 "
Vegetable matter,	9.31 "
Water,	16.87
	100.00 "

* All the analyses given in this paper were made by Prof. H. C. Bowen, of the School of Mines, Columbia College, Chemist to the Barber Asphalt Paving Company.

The earthy matter consists mostly of clay, and the rest is very fine sand. The refined asphalt has the following properties: Specific gravity, 1.49; hardness at 70° F., 2.5; color, black. It breaks with a conchoidal fracture, burns with a yellowish-white flame, and in burning emits an empyreumatic odor. It yields on partial analysis:

	Per cent.
Bitumen (by CS_2),	59.86
Earthy matter,	35.82
Vegetable matter,	4.32
	100.00

And on ultimate analysis of the pure bitumen dissolved out by CS_2

	Per cent.
Carbon (C),	85 9
Hydrogen (H),	11.06
Sulphur (S),	2.49
Unknown, possibly oxygen,	0.56
	100.00

After treatment with petroleum residuum, in order to make asphalt paving-cement, the bitumen soluble in CS_2 varies from 68.5 to 70 per cent.

The Principal Uses of Refined Asphalt.—These are as follows: 1. As a varnish or paint; 2. As an insulating-material; 3. As a water-proofing material; 4. As a cement in ordinary construction; 5. As a cement in roofing- and paving-compounds.

For all of these purposes it is, in its natural state, too brittle at ordinary temperatures. It is, therefore, treated or tempered with some form of oil, the kind of oil and the amount used depending on the purpose to which it is to be applied. For a thin varnish it is mixed with one-third of oil of turpentine and one-sixth of shellac. For paving-cement it is mixed with one-sixth of petroleum residuum. This cement is also suitable for coating water-pipes; but as these are not subject to atmospheric influences, they are usually coated with coal-tar residuum. An insulating material for electric cables, called bitite, is made by the Callender Insulating Company. Its exact composition is not known, but the principal ingredient is asphalt. Asphalt paving-cement is also used to imbed naked copper wires for electric lighting. Mixed with sand, in substantially the same proportions as for paving, asphalt has been used to make electrical conduits on the "drawing-in system." As to the importance of using a non-conducting material in such conduits, in which insu-

lated wires and cables are to be placed, electrical experts are divided in opinion. A serious objection to the use of such material in the streets of New York is the escaping steam from steam-heating pipes, which frequently raises the temperature of the surrounding earth to over 250° F., at which temperature the conduit loses its shape under pressure. Where such pipes are not laid, these conduits are believed to be entirely satisfactory.

In masonry construction, for the reasons stated at the beginning of this paper, bituminous cement is used only in those cases in which rigidity is undesirable and impermeability to water is necessary. A notable instance of this is the brick arch spanning the large wrought-iron water-main on the High Bridge over the Harlem River in New York. It was essential that this arch should be water-proof; and as its length is 1381 feet, a ductile cement was indispensable on account of expansion and contraction. The Chief Engineer (Gen. G. S. Greene) used a cement-mortar, made of Trinidad asphalt with the addition of 10 per cent. by volume of coal-tar residuum and 25 per cent. of sand. The arch was a segment of a circle, of radius 29 feet 6 inches, and span 17 feet, the thickness being 12 inches, composed of three courses of bricks. The first two courses were laid on edge with ordinary mortar of hydraulic cement and sand. On the second course a layer of the asphalt-cement mortar was laid to the thickness of half an inch. On this, while still hot, the third course of bricks was laid flat, the bricks being dry and hot, and dipped in hot asphalt before laying. The joints were then filled with hot asphalt, and finally a course of pressed brick was laid flat in hydraulic-cement mortar, forming the paving and flooring of the bridge. The work is nearly thirty years old, and has never leaked a drop.

Another noticeable instance is the La Salle Street tunnel under the Chicago river, the arch of which was laid with asphalt cement. The tunnel has always been perfectly dry; whereas the Washington Street tunnel, under the same river, the arch of which was laid in hydraulic cement, has always leaked so badly that at times the tunnel cannot be used at all.

For the purpose of preventing dampness in cellars, the same form of construction, using two thicknesses of bricks dipped in asphalt while hot, and separated by a layer of asphalt, is entirely successful.

The usual method of covering arches with a continuous sheet of asphalt is not always successful, as there is an unequal contraction in the asphalt and the masonry with which it is in contact, which results in cracks. By using asphalt as the cement between heated

bricks, this difficulty is avoided, since the contraction in a single joint is almost infinitesimal.

A special form of damp course is also made by saturating hemp, canvas or felt-paper with a mixture of asphalt and very fine sand.

Of the use of asphalt in foundations, two very interesting examples are given by W. H. Delano, in a paper read before the English Institution of Civil Engineers in 1880. One was the foundation of a rock-disintegrator, running at a high rate of speed. It was first built upon a foundation of ordinary concrete. On the opposite side of the street was an establishment for painting on glass and china, where fine grades of work were required. The vibrations from the disintegrator were so great that the business of the glass-painter was rendered impossible. He threatened suit for heavy damages, whereupon the foundations of the disintegrator were removed and rebuilt in bituminous concrete. The result was entirely successful, the vibrations becoming imperceptible. The second case was the foundation of a large trip-hammer, weighing forty-five tons, which was erected at the Paris Exposition of 1867. In order to reduce the concussion, this was built in bituminous concrete, and with entire success. At the close of the Exposition, the concrete was so tough that it was found impossible to make any impression on it with a pick or chisel. As blasting was not permitted, the foundation had to be left in position, and it is believed to be still there.

In all of these cases the concrete was made from the mastic cakes described under the head of Bituminous Limestones in the preceding portion of this paper. The proportions were 60 per cent. of broken stone and 40 of gritted asphalt-mastic (see p. 7). It was tamped between wooden frames, secured by iron cross-bolts, and these bolts were left in the material. An equally good concrete can be made with asphaltic paving-cement, mixed with two parts of sand and four parts of stone in an ordinary concrete-mixer. It is believed that such a concrete, with a thin facing of steel, would make a very tough and durable wall for fortifications.

Of all the uses made of asphalt by far the most important is the paving of roadways.

During the last twelve years, upwards of 3,500,000 square yards of genuine asphalt pavements have been laid in the United States. They extend over a length of more than 200 miles, and are used daily by probably 50,000 vehicles. The cubical contents of the asphalt surfaces are over 7,000,000 cubic feet. With their founda-

tions of concrete, the total contents are nearly 23,000,000 cubic feet and weigh a million and a half tons. The great pyramid of Cheops is not quite four times as large. It is supposed to have occupied the labor of an army of men for one or more generations, and has been the wonder of the world throughout recorded history. It is significant of the resources of the nineteenth century in mechanical appliances that an equal amount of material could now be handled in a few years, in a single industry, without attracting any attention.

The city of Buffalo has now the distinction of possessing more asphalt pavements than any city in the world, though the combined area of the asphalt and the tar pavements in Washington is still greater by about 50 per cent. The area of asphalt here is 1,000,248 square yards, extending over a length of 51 miles, more than the combined area of all the asphalt roadways in Europe.

The form of construction which has usually been followed is a rigid foundation of ordinary concrete, six inches thick, on which the asphalt wearing-surface is placed. In some instances, the foundation is of bituminous concrete. Each form of foundation has its advantages and disadvantages. The ordinary concrete has an excess of strength for the purpose, and the bituminous concrete is doubtless strong enough. With ordinary concrete, the bond between the foundation and the wearing-surface is not very great; hence, it is very easy to strip off the surface in case repairs are necessary; but, on the other hand, the surface sometimes slips on the foundation, and, under traffic, rolls into waves and irregular surfaces, and sometimes cracks with sudden and great changes of temperature. With bituminous concrete, the foundation and wearing-surface are united into one mass and cannot easily be separated. Repairs are more difficult, but waving and cracking are less frequent. The bituminous concrete is less expensive. As it is not exposed to the atmosphere, coal-tar residuum can be used in place of asphalt as the cementing-material, and a strong concrete can be easily made by simply spreading the stone, compacting it with a roller and pouring hot tar over and into it. It is not generally known how strong such a concrete becomes with age. As an example, I may cite a case which fell under my notice in Washington some years ago. I was driving over such a pavement one day, when I noticed a small hole, about the size of a horse's foot. On probing it with a cane, I discovered that I could not touch the bottom. I immediately had the pavement opened, and was astonished to find a cavity about twenty by

forty feet in extent and from four to five feet deep. The earth from this had been washed into a defective sewer; and it must have taken months, if not a year or more, for this to occur, for the hole in the sewer was quite small. During all this time, the pavement was simply a concrete arch, six inches thick, and having a span of about twenty feet; and yet it had sustained the traffic until finally a horse broke a small hole in it. The traffic was infrequent; but at times heavy ice and coal-carts passed over it. The pavement was about ten years old.

Whatever the form of foundation, it is desirable that the surface should be cemented with asphalt without the use of any product of coal-tar. The reason for this has already been given, viz.: that the products of coal-tar are subject to oxidation by the atmosphere, which in time renders them brittle and friable and devoid of any cementing qualities. The asphalt of Trinidad is not subject to this defect, for the reason that it has been exposed for centuries to the burning sun of a tropical climate, and the atmosphere can have no further effect upon it. It is a stable compound, and its cementing-qualities are proof against any atmospheric influences. I am aware that one of the pavements in Washington, in which the cementing-material was a mixture of tar and asphalt, has proved successful after fifteen years' use. But, on the other hand, of other pavements, laid by the same parties, at the same time and under the same specifications, some had to be resurfaced with asphalt after a few years' use, and others are now in a deplorable condition.

With the asphalt as a cement, is combined the proper proportion (about 7 to 1 by weight or 4 to 1 by volume) of fine, sharp sand, ranging from $\frac{1}{20}$ to $\frac{1}{100}$ of an inch in size. The two materials are thoroughly incorporated by suitable machinery at a temperature of about 300° F., and the plastic composition is taken to the street while still hot, spread on the foundation by means of hot iron rakes and compacted by tamping and rolling. The result is a compact, durable sand-stone of fine, uniform grain, with a slightly granular surface, the advantage of which will be subsequently referred to. The bituminous nature of the cement makes the stone somewhat elastic, or, more properly speaking, ductile, and enables it to resist the heavy blows of horses and wheels without breaking. A rigid pavement of Portland cement-concrete, recently tried in New York, lasted just five months before it was removed. The first holes appeared in three weeks.

Of the various compositions for roofing or paving, in which asphalt

Fig. 1.

View of the Pitch Lake in the Island of Trinidad, British West Indies.

Piece of Asphalt Pavement from Linwood Avenue, near North Street, Buffalo, laid October, 1882. Cut out October, 1888.

FIG. 2.

or coal-tar residuum is the cement, the name is legion. A digest of the patents for these compositions, issued about twenty years ago, shows several hundred of them. It is not believed that any of these patents had any value. The virtue of the composition depended on the quality and quantity of the asphalt or the tar used, and all the other substances introduced (except the sand which formed the body) were useless or worse. In the asphalt pavement, the specifications have usually called for a certain amount of powdered limestone, to be mixed with the sand. This was used mechanically, not chemically, for the purpose of filling the voids in the sand, absorbing the free asphalt in the voids and making a more compact and homogeneous compound. All of this can be accomplished by the sand itself, if part of it is sufficiently fine.

I have referred to the granular surface of the Trinidad asphalt pavement. The importance of this is very great in affording a foothold to horses and diminishing the slipperiness which any uniform surface must have to a greater or less degree. By passing the finger over two samples, one of the natural bituminous limestone (or French asphalt) pavement and the other of the artificial bituminous sandstone (or Trinidad asphalt) pavement, the difference is very noticeable. It is the difference between fine sandpaper and polished marble. The difference in practice was shown by careful comparative tests, made in the autumn of 1885, on two pieces of the respective pavements on the same street in New York. Out of a total of over 36,000 horses passing in six days, 5 fell on the Trinidad asphalt (sandstone) and 95 on the French asphalt (limestone). A series of comprehensive tests was made at the same time on different pavements in various cities in the United States, which showed that under all conditions of weather and climate fewer horses fell on asphalt than on stone pavements, confirming the result of similar tests previously made by Colonel Haywood on the streets of London.

The pavements which I have just described are not the only pavements in which asphalt is used. Having an unbroken monolithic surface, they (in common with the pavements having a similar surface, in which coal-tar is used as the cementing-material) have sometimes been called "sheet"-pavements, to distinguish them from "block"-pavements made of asphalt. The latter are made of crushed limestone, in size from ½ inch to dust, mixed with 10 per cent. of asphalt-cement. As the mixture leaves the mixing-apparatus, it passes into a press not unlike a brick-machine, and is there moulded and compressed into blocks 4 x 5 x 12 inches in size.

These blocks are cooled, and can then be handled at any ordinary temperatures. They are laid on the street, in the same manner as stone blocks or bricks, to form a pavement. They have the advantage over any monolithic or "sheet"-pavement of being made at a factory, whence they can be transported to the point where required, and laid by ordinary paviers without the aid of skilled labor; whereas, sheet-pavements require special machinery and skilled labor in each city where they are laid. The asphaltic blocks are also much smoother and less noisy than stone block-pavements, and they are practically waterproof, because the joints are so narrow that, under the sun's heat and the traffic, the asphalt cements the blocks together. On the other hand, they are less durable than the other pavements named.

It has been found impossible to use sand in the manufacture of these blocks, as it cuts the moulds. Hence, limestone is used to form the body of the block; and this wears rapidly under heavy traffic. On residence-streets, however, where travel and traffic are light, they have given great satisfaction; and more than 500,000 square yards, or about 25 miles of them have been laid during the last ten years.

In conclusion, I trust it will not be considered out of place if I call attention very briefly to some of the merits of asphalt-pavements and their effects upon cross-city transportation. I shall not speak of the comfort of driving over a smooth uniform surface, or of the advantage to health of a pavement which has no joints to collect street-manure when it rains, and then blow it into our lungs with the next sunshine and breeze, or of the saving of wear and tear on the nervous system by getting rid of the "mighty roar" of stone pavements. These considerations are quite well understood, and it is generally considered that an asphalt pavement is an agreeable luxury for residence-streets. It is not so well known that the cost of maintenance is a mere function of traffic, and that where the traffic is heavy and the cost of maintenance apparently large, the corresponding saving in traction-force and in wear and tear of vehicles is still greater. The statistics and figures necessary to establish this fact are too long and complicated to be given here. But it is susceptible of direct proof that if we count the cost of paving any large city, like New York, for example, with asphalt, and maintaining it at the cost shown by years of experience under varying weights of traffic, and then count the saving in the cost of transportation and wear and tear of vehicles, it will be conclusively shown that the

saving effected is very nearly three times the cost. This will not appear so remarkable when it is remembered that the transportation through the streets of New York is something over 40,000,000 ton-miles per annum, costing over $15,000,000, and that the repairs of 30,000 vehicles and the shoeing of its 40,000 horses cost nearly $4,000,000 in addition per annum.

The force that draws one ton over a stone block-pavement can draw three tons over asphalt, and the cost of repairs of vehicles and horses can be reduced about one-quarter by the use of smooth pavements. The saving thus effected in both directions runs, as will be seen at a glance, into the millions of dollars per annum. When these facts come to be more widely known, it is not improbable that smooth pavements will cause a revolution in the methods and cost of transportation within cities, second only in importance to the change effected by railroads in transportation between them.

Discussion.

W. H. Delano, London (condensed from a communication to the Secretary): Capt. Greene says that asphalt is a synonym for native bitumen. It may have been so at the time Noah's Ark was built, precision in technical terms being then comparatively unnecessary; but Mr. Malo's nomenclature is now generally received. It has no disadvantage, technical or scientific, and to change it for the behoof of the United States alone would be inconvenient. It was in France that the natural product of fine-grained limestone impregnated with pure native bitumen, called by Malo Seyssel asphalt, was first used,—in a liquid form for footpaths,—long before Malo's time. The latter material was then erroneously called "*bitume*," and that is what the Parisian architects call it still—those who know no better. After this liquid asphalt had got well into use, M. Polonceau analyzed it and propounded the theory that it could be produced artificially, which, however, has never yet been proved. Then contractors began to make mastic with gas-tar, fire-clay and chalk, which mixture, being so much cheaper than asphalts from the mines, with carriage to pay and natural bitumen to use as a flux, nearly drove real asphalt from the market. A name was necessary for this material. It was called "*bitume factice*," or simply "*factice*." Afterwards came the entirely new application of compressed asphalt for roadways; the name given to that was "*asphalt comprimé*." Again, as the trade increased, it was found that the word "*bitume*" caused confusion; so that this term came to be applied to pitch alone.

The nomenclature is, and has been for five and twenty years, as follows:

1. *Asphalt rock*, for Oolite limestone naturally impregnated with bitumen.
2. *Compressed asphalt*, used for roadways.
3. *Asphalt-mastic*, for the blocks (as sold in the trade).
4. *Gritted asphalt-mastic*, for footpaths.
5. *Refined bitumen*, for mineral pitch from Trinidad or elsewhere.

Of late years people have tried to find a substitute for compressed asphalt in an artificial compound of Trinidad pitch, fine sand and powdered limestone; the most successful form being "Grahamite" or Barber's asphalt, so-called. This should be called Barber's bituminous compound. It has several congeners in Europe; for instance, the production of Dietrich of Berlin and Farrington of Paris, etc., none of which have made any way on the continent.

Although the artificial compound known as Barber's may be less smooth than compressed Seyssel, Sicilian, or Val de Travers asphalt to the touch, it is just as slippery in muddy weather. In fact, asphalt *per se* is not slippery, it is the greasy mud on the surface in drizzly weather which causes horses to slip, and which should be washed off or neutralized by strewing sand. Some 7800 square meters (9729 square yards) of Barber's compound were laid in the Rue de Rivoli, Paris, in October, 1886. Not a square inch of it remains; it was relaid about four times, but could not resist the traffic. The result in Berlin is the same. Wear begins on this material as soon as it is laid, by reason to the large proportion of sand, and the fact that it is a sort of half-mastic. It is as hard at the bottom as at the top; whereas natural asphalt is always harder at the surface, even when but half an inch thick. Of course, natural asphalt wears, too; but it stands some of the heaviest traffic in the world. It should not, however, be used on a grade of more than 1 in 30.

CAPT. GREENE (communication to the Secretary): As to my definition of asphalt, to which Mr. Delano objects, I beg leave to refer to the *Encyclopædia Britannica*, Appleton's *American Cyclopædia*, Chamber's *Cyclopædia*, the *Imperial Dictionary*, Worcester's *Dictionary*, Watt's *Dictionary of Chemistry*, Ure's *Dictionary of Arts*, Brande & Cox's *Dictionary of Arts*, Rumpf and Mothe's *Dictionnaire Technologique*, Larousse's *Dictionnaire Universel*, Boussignault's *Annales de Chimie*, Dana's *Mineralogy* and *Geology*, and all other standard authorities, each and every one of which says practically the same thing, defining asphalt to be "native bitumen,"

"mineral pitch," "solid bitumen," "a compact kind of bitumen," "*le bitume solide c'est le véritable asphalte*," or other synonymous words; and each of which also cites the Trinidad "lake" as the principal known deposit of *asphalt*.

Against these authorities stand alone Mr. Malo, Mr. Delano, and a few others interested with them in the Seyssel mines, telling us that asphalt means a certain kind of limestone containing a certain specified amount of bitumen. This is not vindicating scientific accuracy; it is merely fighting for a trade-mark. In the definitions of the preceding paper, I have naturally followed the standard authorities.

As to the relative durability of pavements made from bituminous limestones and those made from Trinidad asphalt, whatever the experience may have been in Europe, experience in America shows that the latter are the more durable. The pavements made from bituminous limestone (Neuchatel rock), which were laid on portions of Fifth Avenue and Liberty Streets in New York, wore out after a few years, and have been replaced with Trinidad asphalt pavement. Plans are now being made for a similar replacement during the present season of the Neuchatel asphalt-rock pavement on Pennsylvania Avenue in Washington. These comprise nearly, if not all, the samples of such pavements laid on roadways in the United States.

THE NORTHWESTERN COLORADO COAL-REGION.

BY G. C. HEWETT, GLENWOOD SPRINGS, COL.

(Buffalo Meeting, October, 1888.)

THIS portion of the State, being the northern half of its Pacific slope, is drained by four rivers, the Gunnison, Grand, White and Yampa or Bear, which, with the Green, flowing south from Wyoming, unite to form the Colorado. These streams, fed by the snows of the Rocky Mountains, fall from an altitude of 10,000 to 12,000 feet, at their sources, to 4000 feet, at the confluence of the Grand and Green in Utah, near the State line.

This area, about 150 miles square, was the scene of great development in the periods subsequent to the Carboniferous; but earlier

formations have meagre representation. The whole geological section is torn and contorted in every period up to the most recent, by the movements that made the Continental divide and many outlying peaks and high plateaux.

A single instance may serve to illustrate the extent of this action. The Silurian limestones are lifted and partially eroded to form a plateau, fifty miles across, between the Grand and White Rivers, 3000 to 5000 feet above their level and fifty miles west of the Range.

The direction of all these rivers was determined by anticlinal cracks or fissures, since eroded to cañons of 2000 to 3000 feet in depth. These streams and their tributaries show abundant evidence of glacial action, to which is largely due the extent of the arable land in upper benches and mesas.

Until 1882 nearly all this area belonged to the Ute Indians. Hence the climate is not accurately known from systematic and continuous meteorological records; but it is probable that the rainfall at the same altitudes is 15 to 20 per cent. greater than on the eastern side of the Range, or, in other words, not far from 18 inches per annum. Since, however, much the greater part of this is in snow, the rivers remain at flood during July and furnish water for irrigation, which the rapid descent of the streams makes inexpensive. The paucity of rainfall has allowed the accumulation of vegetable mould and the gradual melting of the snow nourishes an abundance of natural grasses, which are cured by the dry summer and autumn.

Where the Carboniferous measures are exposed, they have been crushed and twisted between the heavy limestones and quartzites below and the enormous Jura-Trias above, and though they contain coal, it is extremely unlikely that any of it possesses commercial value.

The Jura-Trias red sandstones are 3000 to 5000 feet thick, and their fractures have cañoned all the streams of the Piedmont region at least once in their course. On top of these, separated by the Atlantasaurus shales, is the Dakota sandstone, a very prominent ledge, 50 to 100 feet thick.

In this sandstone is the horizon of a coal-seam that may be of value between the White and Yampa rivers and elsewhere.

Above this ledge are 1500 to 2500 feet of Colorado shales, that, when up-turned, are easily eroded, and have formed a chain of parks below the coal-measure sandstones.

The productive Cretaceous coal-measures are 700 to 1200 feet thick and contain two, possibly three, series of coal-seams.

Above the coal, the softer sandstones and shales of the Post-Cretaceous, about 1000 feet thick, have usually been eroded to gentle slopes.

Over the larger part of the basin, the section is increased by at least 3000 feet of Tertiary measures, containing some petroleum shales that, when fresh, yield, on heating, considerable vapor.

Since the region was outlined by the fractures that determined the courses of the rivers, many basaltic overflows have occurred; and, as these cap all the geologically and topographically high ground, it is probable that the whole basin was so covered. In the bed of the Eagle, above its junction with the Grand, is a lava-flow so recent that, though the bed of the stream was changed by it, the edge is almost intact, and the adjoining partially vitrified shale has not been eroded.

The coal-measures underlie most of this region in two great fields; one extending from the north side of the drainage-basin of the Gunnison to the southern watershed of the Yampa, with the eastern and northern outcrops thrown up to 30° to 90° and the southern flat; the other a part of the Green River, Wyoming, field, extending into this State about 40 miles on the Upper Yampa river.

As far as observed, these coal-seams, where unchanged by flexure with consequent heat and pressure, carry a lignite containing from 5 to 20 per cent. of water; so that it is probable that the better coals will be found only on the narrow east and north margin where the pitch averages 45°. In the southeastern corner of the field, on the headwaters of the Gunnison, the seams are horizontal, but, being in narrow and detached patches, have been locally metamorphosed so that in four miles the same seams are anthracite, coking and dry coal.

Going north from this point, anthracite is found in patches high on the sides of Mount Marcellina and the Ragged mountains. The last peak of the latter, Chair mountain, carries on its northern slope from 9000 feet altitude to 10,000 feet a tongue of outcrop that is proved to be anthracite. Two seams only of the lower series are left from erosion, and but one is possibly of workable thickness.

Five miles north, on the opposing pitch, the seams come up on Rock creek as coking coal; from here the outcrop turns to the west around the topographical basin of Coal creek, caused by the protru-

sion and erosion of the Colorado shales, the coal-measure sandstones dipping outwards in cliffs 1000 to 2000 feet high. In this basin, the coals are extremely fat, and produce a hard, bright sonorous coke.

Going north from this disturbance half a mile, the measures again dip to the west under the influence of the fold that made the anticlinal valley of Rock creek and Roaring Fork, and continue so for 23 miles northward to the Grand River, at an average pitch of 45° and as a ridge a thousand feet higher than the parks of the Colorado shales to the east.

This stretch of outcrop, intrinsically a most valuable part of the field, is cut by three streams that can be used by railway lines, viz.: at six miles from the basin by one fork of Thompson creek; one mile further by another; and six miles north by Four Mile creek. All these gaps have an altitude of about 8000 feet, necessitating eight or ten miles of 4 per cent. grade to get up to them from the Roaring Fork valley, with an altitude of about 6000 feet.

About opposite Glenwood Springs the measures come under the influence of the great uplift of the White River plateau to the north, around which they describe a wide circle for sixty-nine miles, and, standing at 50° to 90°, are known as the Great Hogback.

In that distance the coal-measures are accessible at eight gaps; but the conditions have nowhere been such as to produce a coking coal.

Sixteen miles northeast of the White River gap, at Meeker, under the erosion of the Yampa valley, the outcrop has lost the influence of the plateau, and is flat, with a western direction for thirty-two miles. Here the influence of the Blue Mountain uplift is felt, and the outcrop is bent to the south for twelve miles to the White river; thence it follows the foothills as a hogback to the State line, a distance of thirty-four miles in a westerly course with but two gaps. Near the State line and just south of this hogback on White river, at the mouth of Douglas creek, there is an upheaval similar to that of Coal Basin, but of less violence, and without the conditions to change the character of the coal from dry to coking.

The geographically valuable parts of this 200 miles of outcrop are the eastern and western, and most of the intrinsically valuable coal is in the southern ten or fifteen miles.

The southern outcrop of this field begins on the headwaters of the north fork of the Gunnison, a few miles west of Coal Basin, where the beds are flat and the erosion great; thence following that river,

receding as it falls, it crosses the Grand 15 miles above their confluence and takes its direction into Utah.

In this field, as elsewhere in the State, the Halymenites sandstone is the only datum that approaches persistency. It varies from a compact, hard, white sandrock to a coarse, sandy shale, and over thirty miles of the southeastern outcrop the fossil bed on top of it is quite persistent. But it is not always the base of the coal-measures. At one place, west of the Grand, there are two seams of workable size below it. In fact all the measures of the section lack persistency; and beyond a liability of recurrence of two heavy sandstones enclosing the upper series of beds, a section in one gulch shows no resemblance to the next one a couple of miles away. There is no limestone in the measures, though many of the sandstones are impregnated with lime and gypsum.

The 150 to 200 feet of sandstones and shales above the Halymenites sandstone always contains two and often three or four workable seams; and occasionally two or more come together and form a large and usually dirty seam.

Thorough prospecting on ten miles of outcrop, north of Thompson creek, has shown that no seam remains workable in size a greater distance than two miles, and usually not half that distance. All the seams except one (and that worthless) are nearly as variable in character and marking as in size. These facts, with a cross-faulting every 4000 to 6000 feet of from 30 to 275 feet, make it difficult to say which of these beds are main seams and which splits. Whether the upper part of the measures should be divided is not certain. In many places there are two distinct horizons at which seams recur; but at three points of economical importance considerable prospecting disclosed but one, at six hundred feet above the lower series. Development there shows that this series is more irregular and patchy than the lower series.

It seems altogether probable that the break throwing up the outcrop occurred near the margin of the original field of deposition, because the size and number of the measures is greater where the erosion has been further from that line.

All the seams are coking coal from Coal Basin north twelve miles, or about six miles either way from a point opposite to Sopris peak, six miles distant and the termination of the Elk mountains, a spur of the main range. This peak retains an altitude of 12,800 feet. It is a granitic outbreak, the center of four radiating anticlinal valleys; and the heat from it may have been directly or indirectly the cause

of the coking quality of the coal-seams, they are certainly most "fat" immediately opposite it, and around the Coal Basin upheaval. Also the lower coals are the fatter.

On the northern edge of this influence half a mile of development shows a gradual change from a good coking coal with patches of dry coal to a dry coal that with ordinary bee-hive treatment will barely agglutinate. In another half-mile the same seam is dry. In this transition area, a small cross-fault makes the coal fat for twenty or more feet on either side. The dry seams also present wide chemical and physical changes in short distances; a soft and loosely-bedded coal has in a hundred feet become compact and hard without the intervention of a fault. A couple of hundred feet has reduced the water of combination from 12 to 5 per cent. In seams of coal free from binders of slate or bony coal, the squeezing during upheaval has obliterated the cleavage and often the bedding.

The floor and roof of the seams are generally more or less shelly clay slate that slacks and crumbles on exposure. These facts, together with the pitch, make mining difficult; and extensive development will demand the modification of any systems now in use in the United States.

A PRESENT NEED IN THE ENGINEERING PROFESSION.

BY WILLIAM B. POTTER, ST. LOUIS, MO.

(Presidential Address at the Buffalo Meeting, October, 1888.)

WE have come together on this occasion, as so often before the members of the Institute have met in friendly council, to hear and discuss whatever investigation, observation, and experience during the past months may have developed of value or interest to the many departments of knowledge and practice represented in this association. The member, possessed of the true scientific spirit which this Institute is especially intended to foster, brings here the results of his experience and study unrestrained by fear of giving undue advantage to competitor or rival. He appreciates, moreover, the responsibilities of membership and his duty to the cause, and how much is to be gained by himself from the keen but friendly criticism of his fellow-workers and even by the suggestions that may come of shrewd question or chance remark from those who labor in other but related fields.

On the more formal occasion of the annual meeting the presiding officer, in giving place to his successor, after a year of attention to a great extent directed from the narrower limits of his own specialty to the more general field of the profession and the larger interests of the Institute, contributes the results of his observation from this vantage ground and points to the tendency of progress or the demands for special effort.

But it is not alone to written paper and formal discussion, council meeting or address, that the value of these gatherings is due. There are the benefits that come of personal contact with fellow-members and the quiet exchange of experience and thought; the opportunities for observation of varied practice and conditions; the closer relations of Institute and member with leading workers and the important interests of the land—and with all these there are the informalities of excursion and the good things given and taken on convivial occasions to brighten the more serious hours and give a relish to the whole.

For each and all, then, there may be found in these several meetings of the year some call of duty, some benefit to be gained.

Impressed with a sense of the responsibilities of the position with which I have been honored and the duty of calling attention to whatever opportunities may be offered for advancing the interests of the profession—mindful, too, of the benefits which your suggestion and criticism are sure to bestow, I venture to present in a somewhat informal manner on this occasion a subject which seems to me worthy of your interest and consideration.

During the eighteen years of its existence there has been no one subject which has so much engaged the attention of the American Institute of Mining Engineers as that of technical education. It has furnished material for many a paper and served as text for Presidential addresses not a few, besides calling for special committee report and joint discussion with other and kindred associations. The extent to which this subject has been considered is not only a proof of its vast importance, but an evidence of the earnest interest with which it is regarded by the members of the profession, those occupied with the absorbing cares of business life as well as those engaged in professional work or duties directly connected with technical education itself. Surely it argues well for the increasing prosperity and usefulness of the Institute, and the advancement and elevation of the interests it seeks to foster when the great representative body of experienced and active workers of the profession exhibit such a

regard for the training and development of those who are entering upon the work and who must be responsible for the honor and achievements of our calling hereafter.

To appreciate the vast strides that have been made in this country in providing greater opportunities and better methods for the technical training of engineers one has only to examine the earlier papers on this subject in our *Transactions* and compare the crude and limited conditions therein presented with the facilities now afforded and so well set forth in a late address from a past President of this Institute on American mining schools. From an absolute dependence in the early days upon foreign schools and foreign methods, and the crude beginnings in this country in later times—when rough lecture notes and ancient diagram, a limited routine in chemical laboratory, and cool reception at shop, and mine, and furnace-plant were dealt out to the youth who sought the training of an engineer—we have come to the improved equipment and more searching methods of the present day. The working drawing is now produced and studied in detail, the training of the shop is an essential of the course, the laboratory affords not only a grounding in the principles of chemistry and physics, but experience as well in the practical application of these to commercial operations, and includes within its scope the test and trial of the metallurgist's process and appliance, and of the materials and mechanisms relating to engineering art. Field work and summer school give backing to the theories of lecture room. Ability to do as well as learn is becoming more and more the test of fitness for professional degree.

But although so much has been accomplished, there is much that remains to be done. The tireless energy and restless activity of the present age are constantly making new and more exacting demands. The specialization that marks development in all knowledge and effort calls for a still greater division of labor if the work is to be well done.

The engineer of the past generation was to a great extent a general practitioner, called upon to undertake all cases within the wide range of the profession, for the knowledge demanded was comparatively superficial, and the practice limited to a few set forms of treatment. He has now given place to a multitude of specialists spurred on to secure new and greater developments in special lines of the profession, by an uncompromising competition and the host of new facts and their applications which a crowd of investigators and inventors are constantly pouring upon the world.

The development of technical training to meet these ever new and more exacting demands must keep pace with that of practice and profession, or education will be looked upon in the future, as it has, not without some reason, often been regarded in the past, as the study of ancient history likely to swell the head of the youthful victim rather than enable it to direct a ready hand. To secure such wise and timely development as new exigencies demand requires the untiring and united effort of every one concerned. To the labors of the technical guide and instructor must be added the encouragement and means of obtaining keen practical insight which the actual workers can so well supply, and with these must go the material support and assistance—fitting tributes from such as have been blessed in their endeavors and more or less indebted to the work of the engineer.

It is not, however, for the purpose of discussing the general subject of technical education that I would detain you on this occasion —important though that subject is, and of such interest to one connected with the responsible duties of training the would-be engineer, yet somewhat busied with the more active practice of the profession. It is rather to call your attention to some needed opportunities likely to be felt by such as have by their labors and experience earned a place in the profession, and to report the results of some recent efforts made to meet the growing demand.

There is surely no branch of the comprehensive profession of engineering which calls for so much technical instinct and sound judgment as that relating to the broad fields of mining and metallurgy. In the many other divisions of the profession the bench marks have been more clearly established or the bounding stakes have been set in more limited areas or in fields more fully cleared. The problems that are presented for solution either contain a smaller number of variables or may be made to yield to fewer formulæ, and to such as can be applied with wider limits of application in new cases that arise. There are not many mining engineers but have felt how few constants have been determined in their special branches, and in what general terms these must be applied. It is not only in dealing with theoretical and purely technical questions that one may feel handicapped in his work. Indeed, it is more frequently in the treatment of practical problems that special data are required and the means of determining these fully are so seldom within reach. From the examination to determine the nature and extent of mineral deposit to utilizing the properties of the useful substance this supplies,

there is no step in the many complex operations in which the scarcity of exact knowledge and definite laws does not magnify personal equation, and especially call for the exercise of that sound judgment which comes of keen technical instinct, backed by large experience well profited by. How lacking in detail, for instance, is our knowledge of the laws and their complex operations relating to the origin and development of mineral veins, and yet how important such detailed knowledge to the practical results we seek to obtain. The exact conditions under which rock deposits have been laid down—how they have been moved, what strains have been produced, the origin of these and how they have acted upon each kind of rock; whence the solutions and mineral salts contained; what journeys these have made; what storage room is found and how produced—whether by leaching, fracture, fault, enlargement, shrinkage—and the extent of these, besides the complicated question of the filling of this open ground, the various causes, rates and times of deposition. In a general way we know much of these, but how little of the detail we require for reducing the commercial problem to finished plan or complete estimate.

In the exploitation of mineral ground the lack of detailed knowledge such as this, and of other facts and principles affecting its position, extent and relation to adjacent rocks, is likely to prevent the best advantage being taken of natural conditions in laying out the work and providing for the most economical extraction. With the present means, too, how difficult to formulate the strains and stability of materials met with underground or to determine sufficiently to ensure complete control of imprisoned water, gas, the effects of heat and such conditions as affect economy of work and even the life and health of those who labor in mines.

And further, in the treatment of the minerals produced, whether in preparatory dressing or metallurgical extraction, how much there is to learn of law and fact to enable us to get the needed data in each case on which to base and regulate the plant and process to obtain the best results.

And even when the extracted metal is in hand, how much we grope about in using it and in applying the many properties it has. On the slender basis of a few inaccurate physical tests, for lack of better means, or the crude inquiry which is all that chemistry at present suggests—*ultimate* analysis—neglecting the *proximate composition*, upon which so much the metal's qualities depend—upon

such slender basis and imperfectly digested fact, elaborate structures are developed, necessitating a safety factor, often of huge extent.

There is, of course, a vast amount of knowledge bearing on all these points, as any one in trying to master it would confess, and each day brings some addition to the store, but in the critical investigation of some special case, the limitations of this knowledge are soon made plain.

In these days of rapid progress in all fields of activity, there is a wonderful improvement in the conditions which favor the increase of this much needed knowledge of law and fact. Books of reference, current literature and other special means of reporting the progress that is made in discovery or new application, are coming more and more within the reach of all. Cabinets and museums are increasing in number and growing rich in stores of mineral, rock and fossil specimens and in casts and preparations and such other objects as are found most useful for comparison and study. Physical and chemical laboratories are developing constantly new facilities of special apparatus and better method to spur on investigator and aid the manufacturer.

All these are admirable and contribute in no small degree to the benefit of many. But that which is of especial importance, in these days, to the engineer and the vast field of activity in which he labors, is what might be included, for lack of better term, under the general name of Engineering Laboratory, where materials, machinery, appliances and processes might be tested and experimented with in detail—not with reduced laboratory sample, but on a working commercial scale. It is not meant by this that each individual establishment should necessarily include all the varieties of test here implied, although under favorable conditions such comprehensive duty might be brought under one control. Local circumstances, special needs or advantages would serve as a guide to the wiser foundations on which to build.

I am by no means unmindful of the many laboratories already in existence for the chemical and physical test of engineering and other materials, or of the metallurgical laboratories for trial of ores and their methods of treatment. These all serve useful purposes in their way and are encouraging advances over old-time opportunities. They have contributed largely to the developments of recent years and paved the way for another step forward, but to meet the demands that are now pressing this step should be taken with little delay.

Such laboratories as have been established for making physical tests are either provided with appliances of too limited capacity to yield the most important results, or in the few instances where testing machines of sufficient power are at hand they are not available for use by the profession. The government testing machine at the Watertown Arsenal seems unlikely to fulfil the mission for which it was constructed and is so well adapted, while the only other important machines in the country are to be found at private works where they are mainly used for routine tests to serve as a guide in special work. The laboratories where practical trial and test of metallurgical material and processes may be had are confined to work on a limited scale and are not adapted for the solution of many of the problems that arise.

What we especially need in these days are large public testing works where not merely the unreliable turned-down specimen or trimmed fragment can be tested (which are well enough in many instances as a general guide), but where full-size timber and beam or girder, truss or construction member, large blocks of stone and piers of masonry, as well as the many other materials and structures used by the engineer, may be made to yield a more accurate record of their strength and limits of useful application. In such testing works we should find a full equipment of machinery and appliances especially adapted for testing ores and fuels and other mineral substances and products of manufacture on a commercial scale, by carload or lot of larger size, in all the processes and operations connected with mechanical preparation, extraction or development of power. Here too, if provided with abundant space and good margin of boiler and engine power, new machinery and apparatus could be tested to prove their practical usefulness and the working power required.

Time would fail were I to attempt on this occasion even an enumeration in detail of the manifold advantages and opportunities likely to result from such establishments and the judicious use of them. Every engineer, and especially mining engineer, whatever his training or his field of labor, must, and many from hard experience will, appreciate the practical value, and even necessity, of securing such means of obtaining full knowledge for safer guidance and control. The manufacturer, and corporation, too, and all such as operate or invest in mine, or mill, or furnace plant, or many another enterprise, will be made more secure against loss and disap-

pointment, since we have here the means of reducing wild and hazardous schemes to definite business plan and safe estimate.

To obtain the widest use and best advantage from these engineering laboratories they should, as already indicated, be for public business, operated on a good commercial basis with proper technical control, provided with trained engineers and specialists to conduct the trial and read results aright. Here, then, for test of material, process, or appliance, the engineer or others interested could send their material for proof and practical trial on a working scale, and at fair cost obtain reliable results, sending, if desired, their own experts to watch or guide the work.

With such opportunities as these establishments should provide there is not likely to be wanting the means for profiting thereby. Besides commercial work, calling for more or less elaborate experiment and yielding often new and important data, there would be special investigations in the interest of science and its useful application, promoted by private bounty and by such associations as are engaged in related work, or are benefited by its results.

Having thus briefly presented a suggestion of the means for meeting a much felt want in the profession, I venture to refer, in conclusion, to an effort that has been made to carry out the suggestion and apply the plan in practice.

During the past year an establishment has been completed, and has already a record of eight months of successful practical operation, to which the name of St. Louis Sampling and Testing Works has been given. It includes a full line of machinery and appliances on a commercial scale for the crushing, sampling, milling, and concentration of ores, the washing, coking and steaming-test of coals, and for other related work, by the car-load, or in lots of larger size. Special pains have been taken, in providing and setting this machinery, to secure every possible facility for varying the conditions and methods of operation, due attention being given also to such accessory apparatus as may contribute to more accurate trial and the reliable record of all working parts. Laboratories for assay and chemical work, and for physical tests, are provided to guide and control the work as well as promote investigation. Trained and experienced specialists are connected with each department that the needed intelligence and skill for the most effective utilization of these appliances may not be wanting.

So far the work has been mainly of a commercial character. This has developed to a very encouraging extent, and has been the means

of proving the efficiency of the plant and its usefulness to the public. Even for the short time it has been in operation it would be difficult to estimate the amount of money that has been saved to many who were rushing blindly into the erection of expensive plants, or ignorantly experimenting with one costly process after another on a lavish scale. In the prosecution of this commercial work many valuable data have been obtained bearing upon important metallurgical problems, and special investigations have been undertaken from time to time to make the information thus gained more complete.

In several instances members of the profession have made use of the opportunities offered, and themselves conducted investigations to solve some special metallurgic problem in the interest of clients, or to test on a practical working scale some modification of process or new mechanical device.

In the education of students in metallurgy and engineering these works are performing also a most useful office. Indeed, it would be impossible to over-estimate the value of such advantages as are there offered the students of Washington University for becoming thoroughly familiar with the construction and use of the machinery and processes relating especially to mining and metallurgy. No argument should be necessary to convince members of the profession, or those who have occasion to employ young mining engineers or metallurgists, of the importance of practical training in works on a commercial scale.

In this way, it will be seen, a step has been taken still further in advance in the direction already indicated as important to the development of the profession and the many interests it serves. How far this movement will be successful, or in what measure it will fulfil the mission for which it was undertaken, will depend, in a large degree, upon the manner in which those connected with it discharge the responsibilities of their office and the extent to which encouragement and assistance are given by those likely to be benefited by the work. Much, however, will depend upon the members of the engineering profession, and how far they make use of these and similar opportunities for advancing the interests of their calling.

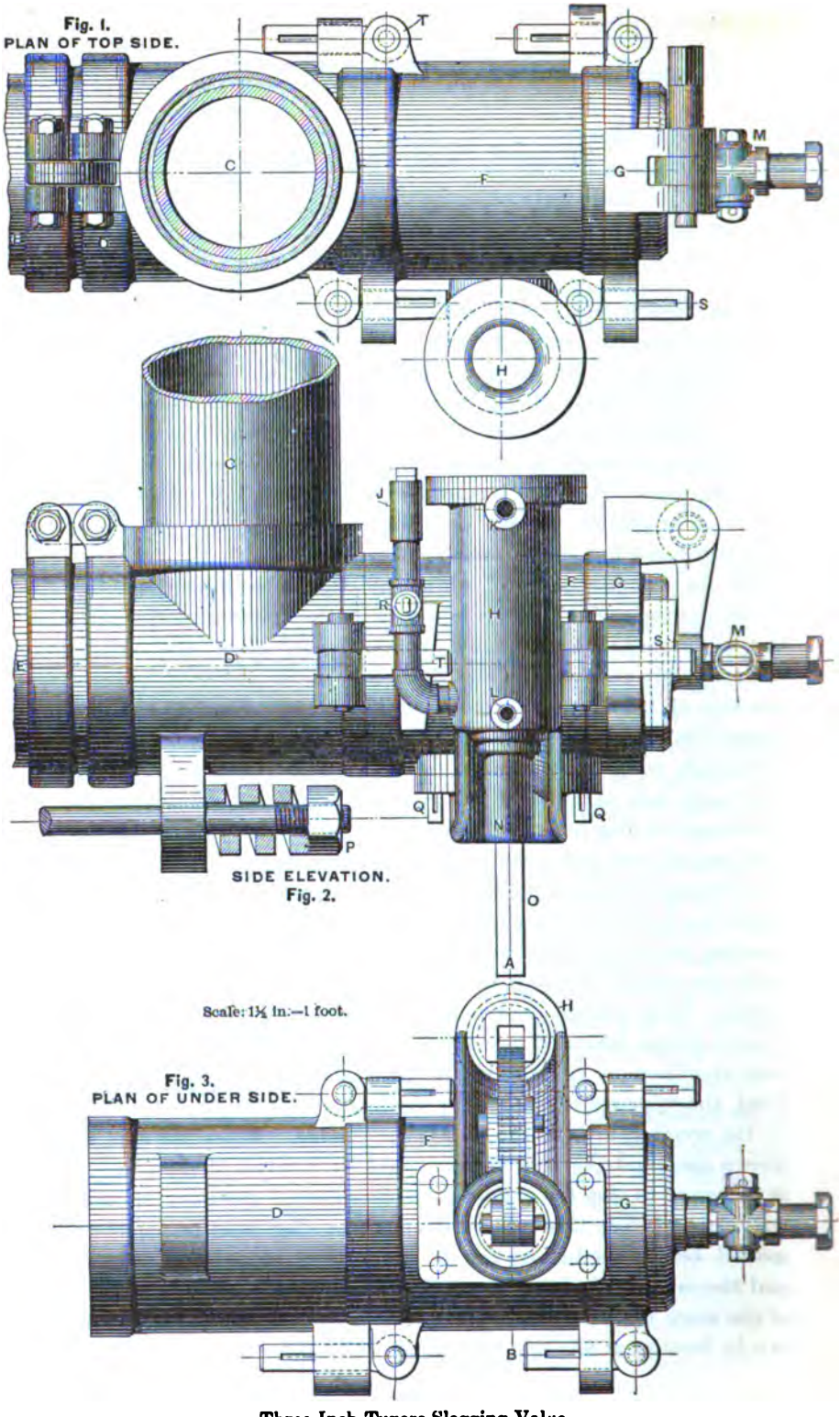
TUYERE SLAGGING-VALVE.

BY EDGAR S. COOK, POTTSTOWN, PA.

(Buffalo Meeting, October, 1888.)

FOR several years past, dating particularly from the days of serious "ore-dirt" complications at the Warwick furnace, I have been desirous of providing some safe, quick and easy method of relieving the tuyeres of slag, when the cinder- and iron-notches are chilled through loss of heat in the crucible or when, from dropping or jumping of stock, the molten slag is forcibly driven into the belly- and leg-pipes. Various methods were proposed, but objections of one kind or another always presented themselves, and the subject was postponed to a more convenient season.

In the summer of 1887, while running on anthracite alone, I found the various expedients in use to be so utterly valueless, that the importance of a practicable tuyere slagging-valve impressed me more forcibly than ever. The holes left in the cast-iron jacket at one side of the bottom of the water-blocks were found to be of no value whatever. The swinging or hinged cap on the elbow of the belly-pipe, while convenient to permit the removal of slag after the belly-pipe was partially or wholly filled, could not be adapted to discharge the slag before solidifying, except at great risk to the men and loss of time just when the constant running of the engine and the keeping of blast on the furnace was of the utmost importance. By removing keys from the caps and holding them in place with props, the slag could be discharged through the tuyeres and belly-pipes, with the engine running, by knocking the props away at a given signal. This method is extremely hazardous to life and limb, and I have always hesitated to make use of it. When the furnace is in such condition as to require the adoption of some method of the kind, the stoppage of the engine, cooling the blast and the contents of the crucible, always aggravates the trouble. When the caps are blown open and the slag is discharged by removal of the props, it is necessary to stop the engine to replace them. An hour or more will elapse before this can be done, as the slag must be partly removed before the tuyeres can be approached by ordinary mortals, and the caps handled and fastened in place again. Any movement of the stock is liable to burn the workman, as he stands, securing the cap in front or at one side of the open belly-pipe.



Three-Inch Tuyere Slagging-Valve.

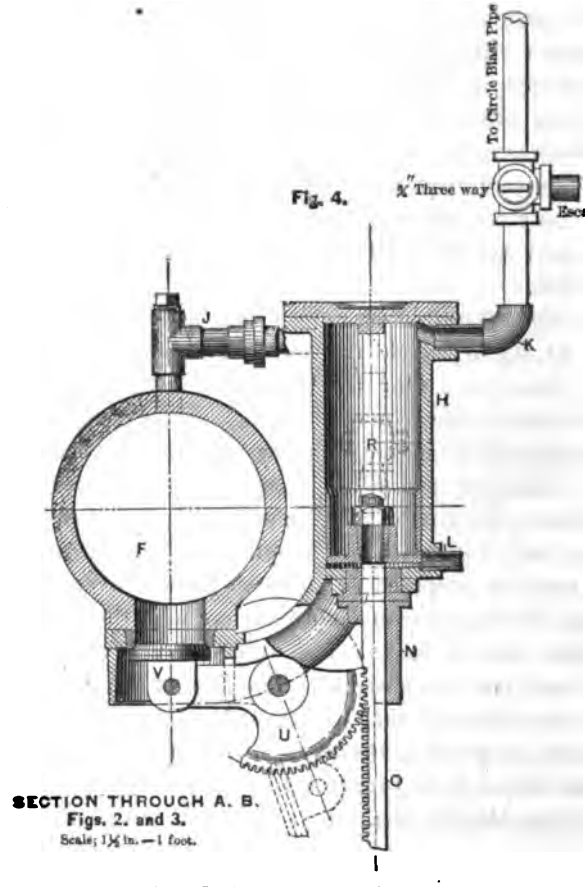
The cooling of crucible-contents and the lowering of the heat of the stoves, while thus standing, adds to the difficulty of restoring the furnace to a normal condition and renders necessary a resort to this method a second, third or fourth time, and even oftener, the conditions growing more unfavorable each hour, and thus frequently causes the loss of the furnace which could have been prevented from chilling, while the work was made safe to all engaged, if there had been a quick, easy method of removing the slag through the tuyeres and belly-pipes without loss of time or heat.

We attempted in 1887 to use the method just described, but our belly-pipes filled solid with slag before we could get props in place, the change in the furnace coming unexpectedly and quickly. Several hours were required to clean the pipes. Fortunately, the trouble was only temporary. On starting up again, the slag cleared the tuyeres and ran from an enlarged cinder-notch without further serious trouble.

This experience brought into existence the tuyere slagging-valve now in use at the Warwick furnace, and found to be valuable in more ways than one. Our chief engineer suggested that a valve placed in the cap or the bottom of the belly-pipe, connected with the piston of a cylinder of larger diameter than the valve and operated with a three-way cock placed in a pipe connecting the cylinder with the main blast-pipe, would afford a safe and sure way of flushing through any belly-pipe without stopping the engine at all. The three-way cock he proposed so to locate at a convenient position that the slag discharged would not interfere with the man operating it. The idea seemed feasible, and orders were given to have one constructed and put into position. The trial of the first valve completed was so satisfactory that we arranged to put similar valves on all our tuyeres, improving the mechanical construction and convenience of the device as we progressed. The accompanying drawings will explain the construction. Figs. 1, 2, 3 and 4 represent the ordinary 3-inch valve.

In these figures, C is the leg-pipe; D, the elbow; E, the belly-pipe; F, the circular casting, keyed to elbow by keys, T. This casting carries the valve-seat, held in position by keys, Q, and is of the same diameter, external and internal, as the elbow, D. G is the swinging cap of the elbow, held in position by keys, S. H is the valve-cylinder, with K opening for connecting-pipe running to the circular blast-pipe and a small opening, L, at bottom, for the escape to the atmosphere of any air leaking past the piston. M is the

usual eye-hole, arranged with a stop-cock to permit the easy cleaning of the glass. N is the guide for the rack-extension, O, of the piston-rod; U, the sector with extended arm, carrying the valve, V, and fitted to work easily with rack, O. In practice, we found that when the valve, V, was thrown open by pressure of air in the belly-pipe (the pressure in valve-cylinder being cut off) owing to its own



Three-Inch Tuyere Slagging-Valve.

weight and that of piston and piston-rod, V would partially close again, not enough to interfere with the flow of slag through the opening, but just enough for the slag to strike it and in time cut it. In order to hold the open valve in the position shown in Fig. 4, a $\frac{3}{4}$" pipe-connection, J, was made from the top of the circular casting, F, to the lower part of the valve-cylinder, H. The opening in

the cylinder is immediately opposite the piston when the valve is closed. When the valve, V, is opened and the piston of the cylinder rises, air passes through pipe, J, to the lower side of the piston and assists to throw the valve, V, wide open and hold it in the position shown. In pipe, J, is placed a stop-cock, R, to regulate the amount of air to be admitted to the lower side of the piston. A brass plug is screwed into top of the tee to facilitate the cleaning of the pipe should it become filled with plumbago or gas-dirt.

The tuyere slagging-valve is easily applied to any of the usual forms of belly-pipes. The connection between it and the main blast-pipe can be made by any one accustomed to put up gas- or steam-pipe. One main pipe, 1½ inches in diameter, is connected with the main blast- or bustle-pipe, and from this ¾-inch branches are run to the several tuyeres, one to each tuyere-arch. A 1½-inch three-way cock is placed in the 1½-inch pipe and ¾-inch three-way cocks in each of the branches, located as may be found most convenient. The ¾-inch pipe of each tuyere-arch is connected by means of a brass union with the top of the valve-cylinder. To keep the valve closed, the furnace being in operation, the plug of the 1½-inch three-way cock is so turned that the opening to the atmosphere is closed and the passage from main blast-pipe to branches and cylinders is opened. The ¾-inch cocks are turned the same way. The pressure of blast in the belly-pipe is the same as in the valve-cylinder. The area of the piston, being larger than the area of the valve in the belly-pipe, the latter is kept tightly closed, never leaking. If it is desired to open a valve, the plug of a ¾-inch three-way cock is turned, cutting off the supply of air from the blast-pipe and opening its passage to the atmosphere. The pressure of blast in the belly-pipe, acting upon the valve, forces it open, driving up the piston of the valve-cylinder, upon which there is now no pressure. The air filling the cylinders escapes to the atmosphere through the side-opening of the cock. To close the valve, the plug of the ¾-inch three-way cock is turned in the opposite direction, closing the passage to the atmosphere and opening communication with the blast-pipe, the pressure from which, acting on the piston of the cylinder, overcomes the resistance of the air in the belly-pipe and closes the valve. By operating the 1½-inch three-way cock, all of the valves in use, one or six, can be opened instantaneously; or each valve can be used separately by operating the corresponding ¾-inch three-way cocks.

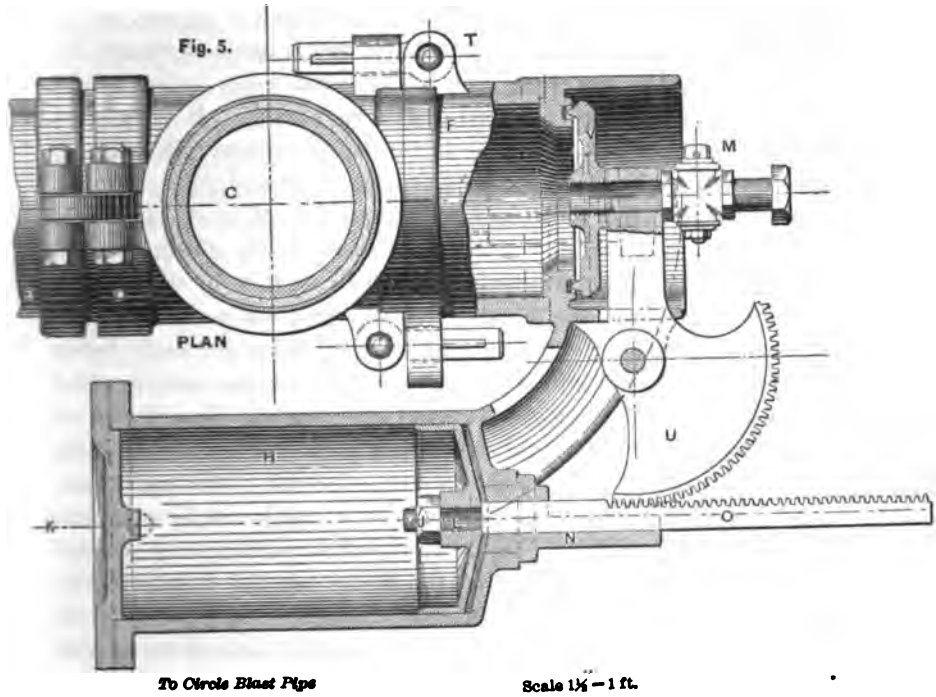
Slag driven into the belly-pipes by the jumping of the stock when the working of the furnace is temporarily deranged, can be quickly

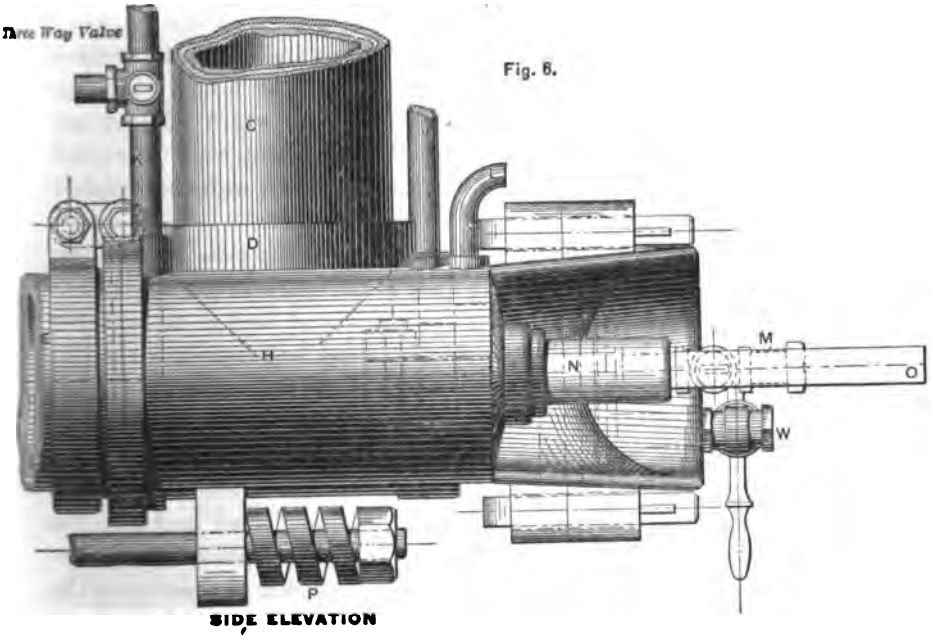
and easily discharged through one or more of the slagging-valves, without loss of time or any inconvenience whatever. When the crucible is badly chilled, the iron- and cinder-notches are closed, and the slag melted by each tuyere is prevented from distributing itself over the area of the crucible, and confined to the immediate vicinity of the tuyere melting it, this slag can be discharged through the tuyere slagging-valves with no risk to the men and without stopping the engine for a moment, thus avoiding all loss of time and all the dangers attending the further cooling off of crucible and blast. The management of a furnace is thus, to some extent, simplified and one of the most serious annoyances attending the operation of an anthracite furnace is removed, viz.: the difficulty of keeping the tuyeres open.

Flushing can be done through the tuyere slagging-valves with the same facility as through the cinder-notch. It occasionally happens, even in a furnace working well, that a pocket forms under or around a certain tuyere, so that when the engine is stopped, this tuyere is sure to fill with slag. When this condition is known to exist, it is only necessary to throw the blast off the furnace, open the valve of the tuyere, put blast on again blowing slag out and then stop the engine.

In addition to the uses mentioned, we have found that the tuyere slagging-valves supersede the "pricking" of the tuyeres, and answer the purpose far better. A pricking-rod is now used only when a large hard lump obstructs the nose of a tuyere, which rarely happens. In order to keep the valves in good working-order, it was our custom to open them regularly once or twice per turn. In so doing, we noticed that the tuyeres kept cleaner and brighter than heretofore. Indeed, we never had such uniformly bright and clean tuyeres, under all conditions of working, as we have had since using these valves—some nine months or more. Besides, we never before made so much iron in the same time. A clean, bright tuyere will certainly melt more material per hour than one furred with dirt. The frequent opening of the slagging-valves removes the dirt deposited in the belly-pipes from the gas, and the infusible dirt that collects around the nose of each tuyere.

We have found a 3-inch valve the most convenient in size, although we have one 6 inches in diameter. This is shown in Figs. 5, 6 and 7, from which it will be seen that in this form the valve, V, replaces the ordinary swinging cap, and carries the eye-hole, M, and the pricker-hole, W, the latter being closed by means of a full-





Six-Inch Tuyere Slagging-Valve.

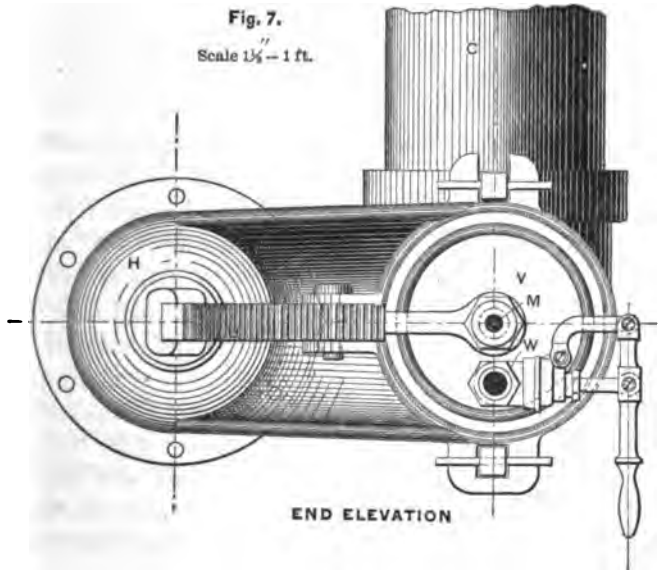
way valve, instead of lever and ball. This valve is located on the front tuyere, directly over the iron-notch, and is made to replace the ordinary cap, the eye-sight and pricking-rod holes being in the valve. It can be easily opened to change nozzles. It is safer to operate a valve of this size independently of the others. Its cylinder is connected to the blast-pipe by a separate 1-inch pipe.

We consider that the 6-inch valve would be of more value than the 3-inch one, chiefly in extreme instances of the chilling of the crucible and a large accumulation of dry ore-dirt in the furnace. As now constructed, the 3-inch tuyere slagging-valve can be placed in position in about five minutes, everything being got ready before the stoppage of the furnace after a cast. A circular casting, about 11 inches long, of the same internal and external diameter as the elbow-casting of the belly-pipe, is keyed to the elbow in place of the cap, the cap being transferred to the opposite end of the valve-casting. In other words, the ordinary elbow is lengthened by a separate casting, 11 inches long. The air-cylinder is permanently attached to this circular casting, carrying the sector and the rack-extension of the piston-rod operating the valve, the valve itself being on the lower side of the casting. The seat of the valve is made detachable, so that it can be renewed, should it become worn or cut by the action of the slag.

Recently we allowed considerable slag to accumulate in the furnace. By simply operating the three-way cock of the 6-inch slagging-valve, we flushed through the belly-pipe and valve without any injury either to the pipe or valve. A very slight shell of chilled slag remained in the pipe. The valve was closed easily and tightly by reversing the cock.

On another occasion, shortly before the regular time for flushing, the engine suddenly stopped, in spite of the efforts of the engineer to keep it moving. A joint in the steam-pipe had been re-made earlier in the day, and we suppose a chip of wood had somehow got into the pipe and caught on the seat of one of the valves of the engine, thus causing its sudden stoppage. The furnace being full of slag, all the belly-pipes, as the engine stopped without warning, filled immediately. The 6-inch slagging-valve opened automatically from the pressure of the slag in the pipe, draining itself and also the adjoining pipe, the only one on which we had not placed a slagging-valve. The 3-inch valves were opened as quickly as possible by the keeper, by means of a small bar, no air-pressure being in the pipes. From each, the slag flowed without difficulty. The delay in open-

ing the 3-inch valves caused a more or less heavy shell to form in the pipes, in contrast with the 6-inch valve, which opened of itself and left but a light shell in its belly-pipe. In half an hour's time all the pipes were perfectly cleaned and we started up again. But for the slagging-valves, the pipes would all have been completely filled with slag, which would have solidified before it would have been possible to remove it by means of the ordinary swinging-caps, thus entailing a stoppage of four or five hours. Such a long



Six-Inch Tuyere Slagging-Valve.

stoppage, coming unexpectedly, might, under some conditions, cause much trouble and additional expense. We think the valves have already repaid to us their original cost.

We are now making blow-off valves for our boilers, constructed on the same principle, with slight modifications to suit the different conditions. Instead of air, steam direct from the boilers is used to operate the valves. Provision is made to guard against the accumulation of water in the cylinder condensed from the steam and to avoid the possibility of the freezing of any of the parts in extreme cold weather.

Letters patent have recently been obtained for the tuyere slagging-valve device.

THE GEOLOGY OF BUFFALO AS RELATED TO NATURAL-GAS EXPLORATIONS ALONG THE NIAGARA RIVER.

BY CHARLES ALBERT ASHBURNER, PITTSBURGH, PA.

(Buffalo Meeting, October, 1888.)

1. The Geology of Buffalo.

The stratigraphical geology of the vicinity of Buffalo has always been of great interest on account of its bearing on the origin and history of the Niagara-river gorge, between the Falls and Lake Ontario. Recently, however, the character and thickness of the strata which underlie Buffalo have acquired more practical and economic importance on account of their relation to the possible occurrence of natural gas in commercial quantity near the Niagara river.

In a paper presented at the Duluth meeting (1887), on "Petroleum and Natural Gas in New York State" (*Trans.*, xvi., 905), I gave a brief description of the four wells which had been drilled by the Buffalo Cement Company between 1883 and that time. One of these wells is 1305 feet deep; and, at the time that my paper was finally revised for the *Transactions* (June, 1888), it was still an open question whether the drill in this well had reached the Niagara limestone, or had penetrated the Medina sandstones and shales, which latter are boldly exposed in a number of localities in the district, but more particularly along the Niagara river in the vicinity of Lewiston and Queenstown.

The difficulty in this geological question at Buffalo has lain in the fact that no reliable determination could be made of the thickness of the strata included between the Corniferous limestone, immediately below which the drilling of the Buffalo gas-wells was commenced, and the Niagara limestone.

On the other hand, the strata included between the latter limestone and the middle portion of the Medina sandstones and shales, have been minutely measured and studied in the Niagara gorge below the Falls by many American and European geologists. The first reliable section of these strata was constructed by Professor Eaton, and published in 1824 in his report on the Geological and Agricultural Survey of the District adjoining the Erie Canal. This section, extending from Lake Erie to Lewiston, is complete, with the exception that no thickness is given to the Salina or Onondaga salt group,

and practically corresponds with new measurements made in 1838 by the now venerable, but still active, State Geologist of New York, Professor James Hall. Hall's section is as follows:

No. VI.	Niagara limestone,	164 feet.
	Niagara shales,	80 feet.
No. V.	Clinton limestone,	25 feet.
	Clinton shale,	4 feet.
No. IV.	Medina sandstones and shales,	350 feet.*

The Niagara limestone forms the knuckle of the Falls, below which 80 feet only of the limestone are visible. The upper part of this formation disappears with a southward dip beneath the surface, about one mile south of the Falls. The Salina formation, which overlies the Niagara formation, occupies all the level country from a point about 2 miles south of the Falls, for a distance of 15 miles by the river, to Black Rock on the northern edge of the city of Buffalo, where the Corniferous limestone outcrops. Within this stretch there are comparatively few rock-exposures, so that it has been impossible to determine the thickness of the Salina strata and any overlying Water-lime or Lower Helderberg limestone which might exist.† The thickness which has been assigned to these strata up to within the past five years has, at the best, been only a guess.

At Black Rock Professor Hall assigned a thickness of 25 feet to the Corniferous limestone and a thickness of 1 foot 2 inches to the Onondaga limestone, which immediately underlies the Corniferous and is frequently included in that formation; but he was never able to determine the thickness of the strata filling the space between these limestones and the Niagara limestone. Later measurements made since 1843, the date of Hall's final report of the Fourth District, have thrown very little additional light on the Niagara-river

* This does not represent the total thickness of the Medina formation, but only that portion which does not extend under Lake Ontario, north of the mouth of the Niagara river.

† Professor Hall claims that the Water-lime, or Buffalo hydraulic-cement bed, does not correspond with the Water-lime group of the eastern part of the State, but with the Magnesian deposit of the Onondaga salt group. The junction of the Water-lime with the Onondaga or Corniferous limestone is plainly visible at several quarries in the vicinity of Buffalo and Black Rock. From a careful study of sections throughout New York I am disposed to believe that Hall's conclusion, published in 1843, that the Water-lime rock in the vicinity of Buffalo belongs to the Onondaga salt group, and that, therefore, the Lower Helderberg and Water-lime groups of eastern New York are not represented in the western part of the State, is correct.

section above the Niagara limestone; of the section in the gorge below the Falls these later measurements have proved the correctness of the general thickness assigned to the strata by Eaton and Hall, and have added much to the differentiation of details, in lacunæ which the greater difficulties of getting at the exposures at an early date rendered it impossible to fill out.

In a recent communication received from Professor Hall, in referring to the Salina group, he says: "In my report no distinct mention was made of the thickness of the Salina formation, since in my original work there was no exposed section in any one place;" and adds, "In the Greigsville shaft, in Livingston county (about 50 miles in an air line east of Buffalo), Mr. Crocker gave a section below the Corniferous to the top of the red shale, as amounting to 458 feet; the red shale may be 300 feet or more thick. In this section he (Crocker) gives as the thickness of the Corniferous limestone and shale 143 feet. My estimate of the prevailing thickness of the Salina formation on the west of the Genesee river is about 800 feet. In one section in Livingston county I was informed, from the boring-records, that from the top of the Tully limestone (part of the Hamilton formation, No. VIII.) to the Corniferous was 603 feet, and thence 995 feet to the Niagara limestone."

Professor Hall, speaking of these records and of others which have been reported to him from the same district, says, "the records do not all agree." While it is impossible for me to discuss, within the limits of this paper, all of these facts, and trace their structural relationship to my conclusion as to the thickness of the strata along the Niagara river, included between the Corniferous and Niagara limestones, I wish to say emphatically that they in no way throw any doubt on my conclusion that the Buffalo 1305-foot well stopped in the Medina sandstone and shale. This conclusion seems to be absolutely proved by the well-borings and geological structure of the eastern end of the Ontario peninsula. If dependence can be placed on the facts reported to Professor Hall, it is quite certain that the Salina strata must thin out very materially between Livingston county and the Niagara river, and that the lower portion of the formation loses its red color.*

It was impossible for me to settle satisfactorily the thickness of

* In referring to this question, in the Fourth District report, Hall says: "The red shale forming the *lower* division of the [Onondaga salt] group, and so well developed in the Third District, I have not been able to find west of the Genesee river. . . . . The red shale has either thinned out or lost its color."

the Salina formation in the vicinity of Buffalo from the record of the deep well (*Trans.*, xvi., 925), because specimens of the drillings, which I might have examined, were not preserved. The strata reported, in the only record of the well which was kept, have been very differently grouped by Dr. Julius Pohlman, the most prominent local geologist in the Niagara district, and by Mr. L. J. Bennett, president of the Buffalo Cement Company; Dr. Pohlman denies that the well, at a depth of 1305 feet, had reached the top of the Niagara limestone, and Mr. Bennett claims that the bottom of the well is 520 feet below the top of the Medina sandstone and shale formation. That the latter claim is substantially correct, I have no doubt.

I have recently made a careful geological examination of the Ontario peninsula between Lakes Erie and Ontario; and the facts obtained during this survey, studied in connection with the records of the three wells recently drilled at Port Colborne (at the southern end of the Welland canal, 17 miles west of Buffalo), enable me to determine the thickness of the Salina strata.

The third well at Port Colborne was drilled 1500 feet. Carefully collected specimens of the strata passed through were kept by Mr. O. C. Richardson, from whom duplicate specimens were obtained. From these specimens I have gathered sufficient facts to enable me to construct a detail section of the well, of which the following is a summary:

		Thickness.	Depth.
	Erie clay, ...	12 feet to	12 feet.
No. VI.	Salina limestone and shales, . .	473 "	485 "
	Niagara limestone, ...	155 "	640 "
	Niagara shales, ...	70 "	710 "
No. V.	Clinton limestone and shales, . .	50 "	760 "
No. IV.	Medina sandstone and shales, . .	740 "	1500 "

The top of this well is 587 feet above tide level; the top of the Medina sandstone was struck at a depth of 760 feet; and, judging from the character of the strata passed through near the bottom of the well, I do not believe the drill stopped very far above the geological horizon between the bottom of the Medina shales and the top of the Hudson-river shales, so that the thickness of the Medina formation is probably not far from 900 feet.

The Clinton limestones and shales are exposed in the Government quarry, back of the Riordan paper mills, at Merritton, 9 miles west of Lewiston. Immediately overlying the Clinton rocks in this quarry is an exposure of the Niagara shales, the bottom of which is 490 feet above tide. The same geological plane (that between the

Clinton and Niagara formations), in the Port Colborne well No. 3, is 123 feet below tide; hence the average southern dip of the strata from Merritton to Port Colborne (17½ miles) is 35 feet per mile.

This result, considered in conjunction with the scattered geological data collected on either side of the Niagara river between Buffalo and Lewiston, and with the records of the Buffalo wells, leads to the conclusion that the bottom of the Niagara shales was pierced, by the drill in the Buffalo deep well, at a depth of 710 feet, and that the top of the Medina sandstone was struck at a depth of 760 feet, so that the drill in this well was stopped in the Medina formation, 545 feet below its upper limit. In this well I would assign a thickness of about 475 feet to the Salina, about 235 feet to the Niagara (both limestone and shales), and 50 feet to the Clinton.

This conclusion is of great importance, since it establishes a new and reliable basis upon which to estimate the depth to which any well would have to be drilled at Buffalo, in order to reach the top of the Trenton limestone, and thus explore for the existence of natural gas, geologically the same as that found in such great abundance, more particularly in sections of Ohio and Indiana. The depth of such a well would depend upon the thickness of the Hudson river and Utica formations, which would have to be drilled through before reaching the Trenton.*

2. Natural-Gas Explorations near the Niagara River.

Since the publication of my paper on "Petroleum and Natural Gas in New York State" I have collected a large number of additional facts, relating to natural-gas explorations at Port Colborne in Ontario, and at Tonawanda and Buffalo in New York, which, on account of the practical attention directed to this subject at the present time, will doubtless prove of interest in this connection.

Gas at Port Colborne.—Four wells have been drilled at Port Colborne, all of which were commenced in the Corniferous limestone, or in the Erie clay which immediately overlies it.

In 1866 a well was drilled a half a mile west of Port Colborne. At the depth of 800 feet the tools were lost, and were never recovered. No oil was found, but a small quantity of natural gas was obtained, and is reported to have been used in a farm-house near by. It evidently came from the upper part of the Medina sandstone.

* Wells are now being drilled at Thorold and St. Catherines, on the Ontario peninsula, to explore the Trenton limestone.

In the village of Port Colborne three wells have been drilled by a number of the citizens, and by the Port Colborne Natural Gas Light and Fuel Company.

Well No. 1 was drilled about 825 feet deep. Gas was first struck at 764 feet, and the sand, reported to have produced gas below this point, was 58 feet thick. Salt water was struck at 452 feet, and was cased off at 455 feet. The gas in this well came from the upper part of the Medina sandstone.

Well No. 2 was drilled by Mr. O. C. Richardson to a depth of 770 feet. Gas was first struck at 752 feet 6 inches, evidently in a higher stratum than that in which the gas was struck in well No. 1. The surface-water in this well was cased off at 168 feet. Salt water was struck below the bottom of this casing, but not shut off. After the completion of the well the water was cased off by a $1\frac{1}{4}$-inch tubing, which rests in a packer 30 feet above the top of the gas-sand.

Well No. 3 was drilled 1500 feet deep, and is still in the Medina sandstone. The principal gas-vein was struck in this well at 763 feet, in the upper part of the Medina sandstone. Small shows of gas were also reported to have been found at 202 feet in a drab limestone; at 278 feet in a greenish shale; and at 413 feet in a drab limestone. A summary of the section of the rocks bored through in this well has already been given. A very small quantity of gas is got from these wells. It is, however, being used for light and fuel in several dwellings in Port Colborne. The drilling of all the Port Colborne wells was badly managed, inasmuch as in no one of them was the salt water absolutely shut off. The water operates very much to the injury of the wells.

Gas at Tonawanda.—Five wells have been drilled in New York State in the vicinity of Getzville, 6 miles southeast of North Tonawanda. The first was drilled about twenty-two years ago to a depth of about 1400 feet. It was drilled wet, and for fourteen years a considerable quantity of water and gas was discharged. The flow of water from the well became a nuisance, and several years ago it was plugged. About the same time two other wells were drilled, but no record was kept of their total depth.

These wells were all drilled to explore for oil, and not for natural gas. The old wells have recently been cleaned out, and the water has been cased off.

Within the past two years two new wells have been drilled, and it is estimated that the total production of all the five wells is about 250,000 cubic feet of gas per day. The wells are within an area of a half a mile square.

The drift which overlies the bed-rock in the vicinity of the wells ranges from 50 to 55 feet thick, and the depth at which the main gas-flow was struck ranges from 420 to 450 feet, although a small amount of gas was got in several wells at a depth of about 200 feet. About 20 feet below this upper gas-vein a salt-water vein was encountered. The Niagara formation contains the reservoir-rock, from which the lower and principal gas-flow is obtained.

The gas from these wells, after being confined for five minutes, accumulates a pressure of 75 pounds to the square inch, which, after continued confinement for from fifteen to twenty-four hours, has risen to 175 pounds per square inch. A company has been organized for introducing this gas into the village of North Tonawanda.

Gas at Buffalo.—The Buffalo Cement Company have drilled eight exploration-wells near Buffalo. Special reference has already been made to four of these in my paper on "Petroleum and Natural Gas in New York State," already cited.

Well No. 1 was drilled with a diamond drill to 490½ feet, and stopped in the lower part of the Salina shales. I have recently examined a complete core preserved of the strata passed through by this well, and they conclusively prove that the lower shales, slates and limestones drilled through belong to the Salina group.

Well No. 2 was drilled in 1884 to 1305 feet. A division of the strata, which I have made from facts communicated to me both by Dr. Pohlman and Mr. Bennett, I have already given.

Well No. 3 was drilled in May, 1887, to 517 feet. Six months after being drilled it produced, in twenty-four hours, 27,600 cubic feet of gas from a pipe 1¼ inches in diameter at a point 865 feet from the well. The pressure of gas in well No. 3 was 60 pounds per square inch fifteen minutes after the well was closed, and the maximum confined pressure at the end of twelve hours was 142 pounds per square inch.

Well No. 4 was drilled to 516 feet, and a considerable flow of gas was struck at 415 feet.

Well No. 5 was drilled to 525 feet, and gas was encountered at the same depth as in well No. 4.

The exact depths of wells Nos. 6 and 7 have not been reported to me, but I understand they were drilled to practically the same depth as well No. 5.

In well No. 7 a fair showing of gas was found at 519½ feet. This well was completed about the middle of October. After the gas was shut in the well it showed a pressure of 65 pounds in fifteen

minutes, 102 pounds in thirty minutes, and 123 pounds at the end of an hour.

Well No. 8 was completed November 14th. The gas in this well showed a pressure, fifteen minutes after being closed in, of 65 pounds; 98 pounds at the end of thirty minutes, and 120 pounds at the end of one hour. A torpedo was subsequently exploded in the gas-rock. Some time after the explosion the gas was shut in the well, and at the end of fifteen minutes it showed a pressure of 100 pounds, at the end of thirty minutes 114 pounds, and at the end of forty-five minutes 119 pounds, when the gas began to leak around the casing, and no effort was made to obtain a higher pressure. Mr. Bennett believes, however, that the pressure of the gas in this well would have risen, at the end of an hour, to 125 pounds.

The total depth of well No. 8 is 560 feet. The principal gas-flow was obtained from the rock between 548 and 554 feet in depth. The strata to 50 feet in this well seem to be more solid than in No. 7. A permanent flow of water was struck at 28 feet, and gradually increased to the depth of 225 feet, where the well was cased and rendered dry.

Wells 3, 4 and 7 were all begun at the bottom of the cement-rock which is quarried by the Buffalo Cement Company, and wells Nos. 5, 6 and 8 were commenced above the same rock.

The tidal elevation of the gas-producing rock in all these wells is practically the same, any variation as to the depth of the gas-producing horizon in the wells being due to the difference in the elevations of the tops of the wells.

The total production of the Buffalo Cement Company's gas-wells to November 1st has not been measured; but the Company had used in their works up to that date and since April 9th, 1888, when the gas was first introduced, 13,032,000 cubic feet of gas. The daily production of the wells on November 1st was 80,000 cubic feet, of which 74,000 cubic feet was utilized for the generation of steam and 6000 cubic feet for heating purposes.

No statement has been made by the company as to the saving effected by the use of natural gas after charging all costs of drilling, etc., over and above the use of bituminous coal, but Mr. Bennett asserts that the natural-gas enterprise of his company has proved a profitable one.

The available heat-units contained in the Buffalo natural gas are practically the same as those of the Wilcox natural gas, which is piped from McKean county, Penna., 85 miles, to the city of Buffalo,

and is distributed throughout the city and sold for 20 cents per thousand cubic feet. The relative value of this latter gas, as compared with anthracite coal for domestic consumption, can be gathered from a report received from a prominent citizen: "My average annual consumption of anthracite coal was 16 tons at $4.50, being $72; my natural gas for one year, November 1st, 1887, to November 1st, 1888, was 297,100 cubic feet at 20 cents, being $59.42. This showing makes 18,569 cubic feet of gas equivalent to one ton of hard coal; but the natural gas account should receive credit for a higher average temperature to which our house was heated, which, of course, I cannot estimate; but I consider it fair to say that we have kept our house, with natural-gas fuel, at an average of 3 to 5 degrees higher than we ever did with coal."

NOTES ON THE ELECTROLYTIC ASSAY OF COPPER.

BY WILLIAM GLENN, BALTIMORE, MD.

(New York Meeting, February, 1889.)

ALMOST beyond doubt, the most important contribution to the assaying of copper yet made, is that of Mr. Eustis (*Trans.*, xi., 120) on the "Comparison of Various Methods of Copper Analysis," which indicates in a manner for which the assayer cannot but feel grateful, how a copper sample ought to be treated from the moment it enters the laboratory until the copper is finally weighed.

Admitting, as the present writer is inclined to do, that this paper establishes, as the official standard, the method of electrolysis, we may note as one of the few details which it does not finally fix, the kind of battery most desirable for precipitating the copper upon platinum. In this respect each assayer appears to be a law unto himself, while all feel more or less "shaky" as to the results. Classen seems to suggest a remedy when he directs the measuring of the gases liberated by the battery employed. But, in fact, that measure is not a constant in the Bunsen elements which he prescribes for copper determination. Furthermore, measuring gases necessitates more machinery and labor than the average assayer has a stomach for. It is, however, but just to Classen, to whom we owe so much, to recall that he wrote more especially for students in the schools.

For some years previous to the reading of Mr. Eustis's paper, the writer had been experimenting with batteries of different types, variously charged, without finding, up to that time, anything better than gravity-cells. Even these, the most constant sources of electrical energy, were uncertain. One could never feel sure, in leaving the laboratory at evening, in what condition the assays would be found next morning. Any considerable fall in the temperature of the laboratory was almost certain to spoil the work, even though the current had been right when the deposition of copper was begun; and it was not rare to have assays fail of precipitation without any apparent reason whatever. It began at last to seem probable that a constant current might be the very thing most to be avoided. It had been evident always that a dense solution of copper gave its metal to the platinum more readily than a dilute one; hence, it might be possible for a constant current to precipitate the metal properly at first, and then towards the end of the electrolysis fail to do so at all, or, at least, do so with intolerable slowness. On the other hand, a constant current might at first set free the copper in a spongy form, while later on, after the metal had mostly left the solution, it might have precisely the proper energy to precipitate copper in the reguline state.

The evidence given by Mr. Eustis's co-workers as to batteries was various, and not entirely convincing. So far as one could detect, no two of the workers used the same battery, charged identically. But as the account given by Mr. Pitkin, analyst for G. H. Nichols & Co., suggested a remedy against constant currents and a method likely to succeed, the writer decided to work upon the lines indicated by him. Messrs. Eimer and Amend supplied the Bunsen cell of 1 gallon capacity and of the following dimensions in centimeters: glass jar, 15 diam. x 20.5 deep inside; rolled zinc, 10 diam. x 18.5 high; porous cup, 7 diam. x 19 high; carbon, 2 x 5 x 22.5 high. This cell has been charged in various ways, at different times. Entirely satisfactory results have been secured with the following solution: 47° Beaumé sulphuric acid (1.48 sp. gr.) 1 liter; water, 1 liter; sodium bichromate, 200 grammes. An equal weight of potassium bichromate may be used, but the resulting salts are more likely than the sodium salts to crystallize in the cell. It will require 500 c.c. of this solution to fill the porous cup completely. Into the outer glass jar, water should be poured to the depth of 7 to 8 centimeters. A single cell, charged in this way, will in one night precipitate, in a clear

reguline form, the copper from one, two, three or four assays. In several years' use, the writer has never seen a single failure.

Within a few months after learning the virtues of a Bunsen cell of the size given and charged in the way named, the writer was in the laboratory of the Orford Copper Company's New York works, and learned from the chemists there that they had adopted the identical cell charged in the same way, only they used Pitkin's solution of hydrochloric acid and potassium bichromate, as set forth in Mr. Eustis's paper already cited. The Orford chemists precipitated the copper on a platinum cylinder, from 60 c.c. of a nitric acid solution of copper, contained in a small beaker. How many assays they had connected to a single cell, I cannot recall, but believe there were six of them. At the moment I saw them, precipitation was complete, and the copper on all the cylinders was bright and clean.

As pointed out by Mr. Mackintosh (*Trans.*, x., 57), it is probably best to use for electrolysis a sulphuric acid solution of copper; and moreover, the metal once precipitated from such a solution may be allowed to remain in the solvent used, and weighed entirely at leisure. This is no small consideration when we recall that the Bunsen cell ceases to act before one reaches the laboratory in the morning. With no current, nitric acid would slowly redissolve the copper.

My own practice is this: upon 2 grammes of any copper-bearing product (except bar-copper), placed in a beaker of 2 in. diam. and $5\frac{1}{4}$ in. depth, is poured at once a mixture of 3 c.c. concentrated sulphuric acid and 17 c.c. concentrated nitric acid. The beaker is then quickly covered, and set upon a cast-iron plate $\frac{5}{8}$ in. thick, under which is a lighted (really good) Bunsen burner, care being taken that the beaker be not nearer the flame than six inches, because, otherwise, sulphur liberated and fused might spoil the assay. When all turbulent action has ceased, the beaker is uncovered, placed four inches from the flame, and permitted to evaporate to dryness. It is then moved immediately over the flame, and left until SO_3 fumes come off freely. One must here be certain that no nitric acid remains, or it may happen, as it did with sample No. 5 of the series given below, that considerable copper is left in solution after electrolysis. After treatment as mentioned, the mass within the beaker will always be found spongy, without sulphur-balls, and at no time will bumping or spirting occur. To the cool mass, about 60 c.c. of water may be added, and the vessel returned to the hot plate, to heat and

boil a half-hour. Filter into a 100 c.c. measuring-flask; make up to the mark; and, by means of a 50 c.c. pipette, transfer in equal portions into two 100 c.c. beakers. Electrolyze both solutions, using cylinders 3.3 by 3.8 cm., well covered by the solutions, made up with water. For the other terminal, employ preferably a cylinder of 2 cm. diameter.

Of course, one may electrolyze but a single solution; but in that case the flask and pipette must be in agreement, and the measuring accurately performed.

If the work be important, after the copper-plated platinum cylinders are withdrawn for weighing, both residues should be poured together, heated to boiling, removed from the flame and treated with a rapid stream of H_2S.

The precipitated sulphide may be filtered off, ignited in the filter wrapped by a wire, the ash dissolved in 2.5 c.c. nitric acid, made alkaline by 5 c.c. ammonia water, filtered and made 10 c.c. with wash-water. The copper is estimated with KCy of the strength of 6 grammes to the liter of water. Below are given five determinations from a random page of the writer's note-book, showing what was found on the two cylinders, and what in the two residues combined.

No.	Kind of Material.	Copper on No.1 Cylinder.	Copper on No. 2 Cylinder.	Found in Residues of Electrolysis.	Per cent. of copper in Sample.	Quantity used for Assay.
1	Bar-copper.....	.7352	.7335	.0007	97.78	$\frac{1}{10}$ of 15.028 grammes.
2	Matte.....	.2221	.2216	.0006	44 42	1 gramme for both assays.
3	Sulphide-ore.....	.1204	.1212	.0005	12.10	2 " " "
4	" "	.1210	.1212	.0010	12.16	" " " "
5	Matte.....	.4573	.4562	.0033	45.84	" " " "

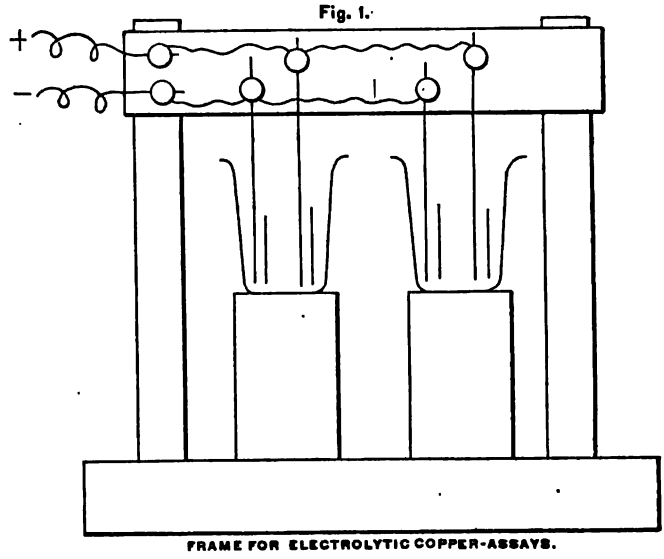
As already observed, the solution of sample 5 contained nitric acid.

Certainly, bar-copper cannot be treated by this method. A large quantity of the properly broken down sample should be dissolved in nitric acid, a known part of this evaporated with slightly more than the quantity of sulphuric acid theoretically required, and the sulphate solution electrolyzed. In the sample above noted, nearly 1.47 grammes of copper were precipitated by the single cell. It would have precipitated equally well two grammes, and so one is at liberty to use that much bar-copper for an analysis, when the method here recommended is used. In this, as in some other laboratories, the

last person to leave at evening charges the battery. The morning following, the cylinders are ready to be weighed.

Some time in 1883, Mr. Kiddie and Mr. Hedley, two of the chemists at the Orford Company's New York works, informed the writer that they had come to mistrust, and had discarded, platinum dishes for electrolytic work, because they had found that nitric acid solutions of copper compounds, even after careful filtering, would sometimes deposit ferric oxide (or some other oxide) after many hours' standing. These oxides would be weighed in the dishes and reckoned as copper. For the same reason, the writer had already abandoned dishes, which he had on hand, and adopted cylinders, which he was obliged to buy —thus unconsciously following in the footsteps of the two chemists

Fig. 1.



FRAME FOR ELECTROLYTIC COPPER-ASSAYS.

quoted. Moreover, this behavior of nitric acid solutions, as well as the work of Mackintosh before cited, had led him to the use of sulphuric acid solutions, which do not appear to deposit anything upon long standing. Finally, necessary evaporation from dishes cannot but expose to oxidation a part of the copper precipitated upon them, while cylinders are not exposed to the air from this cause or any other.

Precisely why a single Bunsen cell, charged as directed above, will precipitate in a bright reguline form a quantity of copper however small, or however large up to two grammes, is not known to me. As

a matter of opinion only, I should say it is because the current is weak while the copper solution is densest, and thenceforth increases at a rate somewhat commensurate with the decrease of copper in the solution under treatment. Be that as it may, the fact remains.

Since neither switches nor resistance-coils are required in the method under discussion, a simple stand, as shown in Fig. 1, is sufficient to hold the platinum in position.

The frame here represented in ¼ its natural size, is of pine. The two posts, 1 in. square, are mortised into the foundation, which is 8 x 12 x 1½ in. thick. The board at the top is 1¼ in. wide and 10 in. long. Into this are screwed binding-posts (represented by circles), which are connected in two series by a wire soldered to each set. It is just 9 in. from the top of the board down to the foundation. The blocks supporting the beakers are 2 in. square and 3½ in. high. The platinum wires should be protected by bits of flattened brass wire, from the marring effects of binding-screws. The frame may be lengthened to accommodate several assays.

To wash the copper, have ready three small beakers of water and one of alcohol. Slip out a block, remove the beaker and quickly wash the copper by means of the beakers of water. Remove the cylinder from binding-post and set it in the alcohol. Drain off the alcohol, ignite the remainder, cool and weigh. No dessicator is necessary.

A RAPID METHOD FOR THE REDUCTION OF FERRIC SULPHATE IN VOLUMETRIC ANALYSIS.

BY CLEMENS JONES, HOKENDAUQUA, PA.

(New York Meeting, February, 1889.)

THE difficulties attending the reduction of ferric sulphate in the determination of metallic iron by the method of Marguerite, are often serious, and affect time, patience and accuracy. They depend upon the condition and purity of the zinc used, the state of saturation of the solution, acidity, and temperature.

D. J. Carnegie (*Chem. News*, lvii., No. 1481), says that ferric solutions are instantaneously reduced by the use of zinc-dust. But there are obvious objections to its employment in quantitative work. Dr.

Drown found that "a concentrated solution, not too acid, was reduced by 5 grammes of pulverized zinc in two or three minutes, without other heating than that due to the chemical action." But the zinc in excess of the stoichiometrical amount must be dissolved. This has been the main obstacle. Large excess of sulphuric acid, heat, or both, must then be employed; and either involves loss of both time and accuracy in the volumetric method.

Marguerite's method is, in point of celerity and refinement, the most scientific of all the "iron methods." If the iron solutions have been properly prepared, the end-reaction is sharper and more quickly obtained than by any other. There is no loss in testing for final oxidation, as potassic permanganate is its own indicator. Besides, the permanganate solution can be most quickly prepared, and accurately standardized.

Hence there is a natural preference for this method in determining metallic iron. It permits direct and rapid titration. Only facilitate the reduction of the ferric sulphate, and it is the most valuable of all methods, combining the utmost rapidity with the highest accuracy attainable.

There is no difficulty in obtaining iron as ferric sulphate; but the reduction of this to the ferrous state has hindered the precision of the permanganate method. Discrepancies in results are all to be set down to the errors introduced in the reduction with zinc. The care to secure complete reduction—how frequently is it lost through the presence of a minute grain of zinc, washed down from the neck of a flask in titrating, and how derisively tossing up a little chain of hydrogen bubbles! Again, a very hot solution will decompose an excess of reagent. And moreover, as I shall again remark, it is probable that the largest impurity in pulverized zinc is not iron, as heretofore supposed, but carbon.

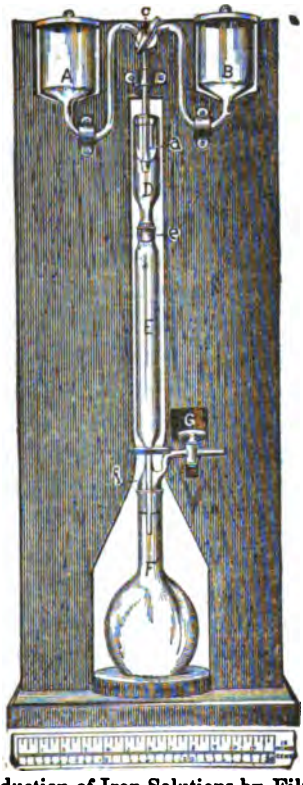
The size of the particles of the pulverized zinc employed in the method which this paper describes, is determined by passing through a 40-mesh, upon a 60-mesh sieve.

Observing the action on zinc of ferric solution in sulphuric acid, I noticed that in all cases, with practically the same amount of acid and zinc, the latter dissolved more rapidly when the solution was more highly saturated with ferric salt; that the time for reduction, under the conditions mentioned, was dependent on the strength of the ferric solution. This was evidence that a sufficient number of zinc particles should be exposed, in a proper ratio to the ferric salt; that each molecule takes the same time to reduce, when unaffected

by its unreduced neighbor, which is the time of contact; and that, therefore, were each molecule presented to new particles of zinc successively reduction would be instantaneous. Filtering the ferric solution through pulverized zinc would then demonstrate this fact, which I found to be the case: *A solution of ferric sulphate is instantaneously and completely reduced by filtering through pulverized zinc.*

The apparatus figured below was designed for the rapid reduction of ferric solutions. It consists of two cups,* A and B, of about 300

FIG. 1.



Reductor, for the Reduction of Iron Solutions by Filtration through Zinc.

c.c. capacity, the outlets of which connect alternately, by means of a 3-way cock, C, into the joint-tube, D. This tube is provided with a packing space, *d*, which is filled with an asbestos roll, kept in place by a ring of cork. The tube moves up and down freely. At

* I find this form of cup not so convenient as the bulb-shape of a separatory funnel of the same capacity.

its lower end, *e*, it is ground to fit the detachable tube, E. This is for the pulverized zinc, and at its lower end has a stopper, *f*, ground to fit the titrating flask, F, of about 700 c.c. capacity. The stop-cock, G, connects with the vacuum-pump. The tube, E, is disconnected by pushing it up, grasping the neck of the flask with one hand, removing with the other the round block which supports the flask, and then detaching the flask, F. The tube, E, now rests on the little shelf shown in the cut. This is drawn out, and frees the tube at its ground-joint, *e*.

A wad of fine glass wool, sufficiently thick to make a good filter, is inserted into the tube, and the pulverized zinc is filled, over this, for four-fifths the length of the tube. This holds about 300 grammes, which is enough for 60 filtrations. The tube is then replaced by reversing the manner of removal, connection is made with the filter-pump, and the apparatus is ready for use. It is first necessary to wash the column of zinc with dilute sulphuric acid. This acid is made up of one part of acid of 1.84 sp. gr. to two parts of distilled water, and is used throughout in all work upon iron determinations. Of this about 30 c.c. is taken, transferred to the cup, A, and diluted to 300 c.c. Stop-cock, G, is opened for suction, and then the three-way cock, C, is turned to discharge the cup, A. A vacuum of 5 to 6 pounds or more, gauge-indication, must be used, since the large volume of disengaged hydrogen retards the filtration. The cup and the column of zinc are then washed free from acid by five rinsings. This operation is repeated four times, and is then finished for this tubeful of zinc.

Below is a series of trials, undertaken to determine the proper correction for iron in the zinc. Owing to the small amount of reagent required for the end-reaction, it was impossible to obtain a reading under 0.1 c.c., which, as will be seen further on, is too much for even so large a quantity of zinc.

No. of Test.	Amount of $K_2Mn_2O_8$ c.c.
1	0.4
2	0.2
3	0.13
4	0.12
1	0.4
2	0.2
3	0.15
4	0.13

No. of Test.	Amount of $K_2Mn_2O_8$ c.c.
1	0.4
2	0.15
3	0.13
4	0.11
1	0.4
2	0.2
3	0.13
4	0.1

These were made at different times upon new quantities of zinc, and with solutions of $K_2Mn_2O_8$, of slightly varying strength. Beyond the fourth trial in each case, it was impossible to get less than one drop of the reagent into the solution. This is nearly 0.1 c.c. and in every instance gave a strong coloration.*

Evidently the largest amount of iron in the zinc is only on the outside of the particles, as the quantity dissolved in the reductor is less than five grammes for each filtration. By the ordinary method of dissolving all the zinc, the correction is rarely below 0.15 c.c. for 5 grammes.

Exactly the same procedure is adopted to reduce a ferric sulphate solution, except to use both cups, A and B. After the first is emptied it is washed with cold water five times. This incidentally cools the column of zinc and retards unnecessary solution. The other cup is then connected by turning the 3-way cock, C, and washed in like manner. In each case this suffices to free the apparatus from all traces of iron or sulphuric acid. G is then closed, the 3-way cock is turned again to relieve the pressure, and the flask is removed. Time for reduction, 2 minutes.

Solutions obtained in this way are beautifully clear and perfectly reduced. The volume and temperature are practically constant: the former being about 400 c.c., and the latter about 40° centigrade. Uniformity marks the entire process, and there is a minimum quantity of zinc sulphate. Dilute solutions must be used. There should be not over 50 c.c. sulphuric acid in 300 c.c. of the ferric solution ready for reduction.

An experiment was tried in reducing the solutions, after oxidation, used in standardizing the permanganate. Iron wire taken at .996 Fe.

Standardization.

	Weight of Wire, grm.	$K_2Mn_2O_8$ c.c.	Reaction.	Factor.
No. 1.	0.1271	16.48	Tinged	1 c.c. = .007681 Fe
No. 2.	0.12175	15.78	Sharp	1 c.c. = .007684 Fe

* I have since obtained a reading down to 0.03 c.c. by closing the burette-tip.

Reduction of these Solutions.

	Reduction.	$K_2Mn_2O_8$ c.c.	Reaction.
No. 1.	complete	16.42	Tinged
No. 2.	"	15.8	Sharp

This treatment applied to a hematite ore, gave (No. 2969):

	Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$ c.c.	Reaction.	Iron, Per cent.
No. 1.	0.1527	complete	7.2	Tinged	34.7207
No. 2.	0.1526	"	7.2	sharp	34.7441
(Reverted: filtered through the Reductor.)					
No. 1.		"	7.2	sharp	
No. 2.		"	7.2	"	

Direct determination of metallic iron in the wire used in standardizing gave the following results: weight of wire, 0.12405 grm. dissolved in sulphuric acid and oxidized. Reduced by filtration.

Reduction.	$K_2Mn_2O_8$ c.c.	Reaction.	Iron, Per cent.
Complete	16.07	sharp	99.5420

Factor, 1 c.c. $K_2Mn_2O_8 = .007684$ Fe.

Solution again reduced by filtration. Reduction complete. $K_2Mn_2O_8$, 16.10 c.c., as compared with 16.07 in the former reduction. Difference, 0.03.

The *apparent* correction for iron in the zinc is the increase in last reading, or 0.03 c.c. of the reagent. Time required to reduce and titrate this solution twice, *ten minutes.*

While the apparent correction is thus 0.03 c.c., it is inadvisable to use it as a constant, since the true correction can be applied for each case, and is really variable, depending upon acidity and degree of saturation.

In the following example, the test of a hematite-ore (No. 2922), carried through in duplicate, the ferric sulphate was first reduced by the usual method of adding the zinc to the solution, and then entirely dissolving it, and deducting 0.3 c.c. as the correction after titration. Each solution was then filtered through the reductor, and on the supposition that for each treatment with zinc (on account of filtration), but 0.1 c.c. is to be deducted; the results are seen to be coincident. By "Reverted" is meant filtered through the reductor. The value of 1 c.c. $K_2Mn_2O_8$ is .007678 Fe.

	Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$	Correction.	Iron, Per cent.
No. 1.	0.1360	complete	7.4 sharp	0.3 = 7.1	40.06
"	Reverted	"	7.3 "	0.2 = 7.1 =	40.06
No. 2.	0.1379	"	7.49 "	0.3 = 7.19 =	40.03
"	Reverted	"	7.39 "	0.2 = 7.19 =	40.03

Another (No. 2971), with a difficultly-fusible residue, was tried directly, each solution being reduced twice by filtration. Time for reducing and titrating in duplicate, twenty minutes.

	Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$ c.c.	Reaction.	Iron, Per cent.
No. 1.	0.1505	complete	9.27	sharp	
2d trial,		"	9.21	"	= 47.0230
No. 2.	0.1823	"	11.29	"	
2d trial,		"	11.23	"	= 46.9132

Assuming that for ordinary purposes the correction obtained by reverting the solution used to standardize the permanganate would be a safe one, I have applied it to the determination of iron in an ore with the following results:

Standardization.

	Weight of Wire, grm.	$K_2Mn_2O_8$ c.c.	Reduction.	Correction c.c.
No. 1.	0.11915	15.48		
"	Reverted	15.55	complete	0.07
No. 2.	0.1261	16.4		
"	Reverted	16.44	"	0.04

Factor, 1 c.c. $K_2Mn_2O_8$ = .007664 Fe.

Determination.

(No. 2997)	Weight of Ore, grm.	$K_2Mn_2O_8$ c.c.	Correction.		Iron, Per cent.
No. 1.	0.1576	9 58	0.07	9.51	46.246
No. 2.	0.2351	14.3	0.07	14.23	46.388

In like manner a rich hematite-ore (No. 3066) gave:

No. 1. Weight of ore, 0.2302 grm.; Metallic iron, 60.03 per cent.
No. 2. " " 0.1645 " " " 60.17 "

Theoretically a neutral or nearly neutral solution of ferric sulphate should be reduced by zinc in this way without dissolving the latter appreciably, at least, not to the extent of introducing the error due to its impurity. As nearly satisfying these conditions, by the use of a minimum of sulphuric acid, the two following examples afford a remarkable demonstration. The experiments were conducted like the foregoing. Factor 1 c.c. $K_2Mn_2O_8$ = .007656 Fe.

No. 1. Weight of Wire, 0.12085 grm.	Amount of $K_2Mn_2O_8$ c.c.
1st trial,	15.72
2d trial,	15.78

No. 2. Weight of Wire, 0.11315 grm.	$K_2Mn_2O_8$ c.c.
1st trial,	14.695
2d trial,	14.695

Metallic iron 99.432 per cent.

There appears to be a deeply-rooted belief that iron is the only impurity in pulverized zinc which reduces the $K_2Mn_2O_8$; but I think the error is in nearly all cases largely due to the carbonaceous residue. Five grammes of zinc were dissolved in sulphuric acid, and then found to require 0.13 c.c. of $K_2Mn_2O_8$. After oxidation, a test of the solution gave *no* coloration with potassium sulpho-cyanate. Oxidation with nitric acid gave a like result.

Twenty-five c c. of sulphuric acid were diluted to 300 c.c., and passed through the reductor. $K_2Mn_2O_8$ required 0.05 c.c. One drop more was added, and the solution allowed to stand until clear. It was then tested with potassium sulpho-cyanate, and gave *no* coloration. A like experience followed in repeated trials of dissolving the zinc directly; but subsequent treatment of these solutions with ammonia and filtration gave evidence of iron. A determination of iron in the zinc used in these experiments gave the following: 5 grammes of zinc were dissolved in sulphuric acid. $K_2Mn_2O_8$ required 0.1 c.c. = 0.0154 per cent. iron. As in the first case, 5 grammes were dissolved and the solution filtered through the reductor. $K_2Mn_2O_8$ required 0.12 c.c. Increase, 0.02 c.c., equivalent to 0.0030 per cent. iron.

Mr. P. W. Shimer has since suggested to me the probability of iron or manganese existing as some compound in the black carbonaceous residue. This view seems confirmed by the fact that ignition of the residue leaves a small fused mass, which, on dissolving in hydrochloric acid, reveals iron by the sulpho-cyanate.

May not this residue be soluble under accidental conditions when exposed to the action of the solution for an hour or more by the old method of reduction? I have frequently noticed that it is perceptibly smaller in bulk. Again, if the solution is warm, may not final oxidation of the ferrous salt have an effect in dissolving it?

If it should prove to be a carbon compound, the reducing action on the $K_2Mn_2O_8$ would be increased beyond the correction for iron as ordinarily obtained. Filtration undoubtedly obviates these difficulties.

As touching this reducing action, the following experience may be cited: An analysis in duplicate by the old method gave an unaccountable difference in results. The one solution, with the high result, was noticed to have a dark color and lustre. Subsequently this proved to be due to a very finely divided sediment, apparently carbon.

(No. 2978) Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$ c.c.	Correction.	Iron, Per cent.
No. 1. 0.2174	complete	11.75	0.3	40.47
No. 2. 0.1795	"	11.12	0.3	46.31
No. 2. Reverted	"	9.61	0.15	40.49

Factor, 1 c.c. $K_2Mn_2O_8$ = .007684 Fe.

Here is another similar case:

Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$ c.c.	Correction.	Iron, Per cent.
No. 1. 0.1951	complete	16.95	0.18	65.80
No. 2. 0.1928	"	15.6	0.18	60.92
No. 1. Reverted	"	15.9	0.3	61.21

Factor, 1 c.c. $K_2Mn_2O_8$ = .007656 Fe.

The following test of an ore is added as an illustration of the capability of this method for rapid reduction. 1 c.c. = .007671 Fe.

Weight of Ore, grm.	Reduction.	$K_2Mn_2O_8$ c.c.	Correction.	Iron, Per cent.
No. 1. 0.1566	complete	13.35	0.05	65.1496
No. 2. 0.1820	"	15.51	0.05	65.1569

Time required to reduce and titrate in duplicate, twenty minutes.

The investigation of this principle of reduction, which I believe to be new, has been governed by the time afforded in the few and brief intervals of regular technical work. I think the method capable of still further advancement and application. At the present stage of the inquiry, it presents the most rapid and accurate treatment for the estimation of iron.

The apparatus was made by Emil Greiner, 63 Maiden Lane, New York, whose intelligent and skilful workmanship has been of great service to me.

BIOGRAPHICAL NOTICE OF ERICH C. SCHAUFUSS.

BY J. H. BOWDEN, WILKES-BARRE, PA.

(New York Meeting, February, 1889.)

ON the night of January 23d, 1889, Erich C. Schaufuss, a member of the Institute, while engaged in making the regular mine-surveys in No. 4 Slope of the Susquehanna Coal Co., at Nanticoke, Pa., was killed by an explosion of fire-damp. The catastrophe, which cost another life also, is attributed to the carelessness of the "fire-boss" leading the party, who advanced with a naked lamp into a chamber

where a body of gas had accumulated.* The result was apparently rather a rush of intense flame from the igniting gas, than an explosion such as is produced by setting fire to dangerous mixtures of gas and air. But to two of the party, Mr. Owen, who was killed by the flame, and the subject of this sketch, who was suffocated by the after-damp, it proved swiftly fatal.

Mr. Schaufuss was born in Dresden, Germany, June 23d, 1856. His father was an eminent jurist, having obtained the honorable position of *Gerichtsrath*. He received a thorough education at the gymnasium, and afterward at the University of Munich, from which he graduated as an architect. While at Munich he was an officer in one of the royal Bavarian regiments. In 1879, Mr. Schaufuss came to this country, and, after remaining in New York for about a year, entered the service of Coxe Bros., at Drifton, Pa., where he worked as a member of an engineer corps for two years. In June, 1882, he removed to Wilkes-Barre, and was employed by the Susquehanna Coal Co., with which he remained until his death. He had risen, through faithful work, to a responsible position, and was recognized as one of the most efficient engineers of the Company. Brave almost to rashness, he never hesitated to undertake the survey of the most dangerous workings.

He was unmarried, and had no relatives in the United States. In Germany, his parents having died, a step-mother and several married sisters constitute the family circle which mourns his death as a stranger in a strange land. But even the sorrow of his kindred cannot be more genuine and sincere than that of his American friends and colleagues, as evinced by the large attendance of engineers and others at his funeral.

He was elected a member of the Institute in 1883, and took a great interest in the meetings, the last of which, at Buffalo, he attended in person. Two weeks before his death, he was planning to revisit Europe next summer, with the party of engineers which will comprise so many of our members.

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* As a matter of justice to the accused person, and a wholesome warning to engineers, the following extract from the report of the Coroner's inquest, given in a Wilkes-Barre paper, is here inserted:

"George Elmey, the day fire-boss, testified that he obeyed his orders, telling John Rudge, the night fire-boss, to accompany the engineering party. He added that he had cautioned Rudge to use particular care in the sloping breasts, such as that in which the explosion occurred, since he considered them particularly dangerous in case the air-currents became disarranged.

"Three of the engineering party testified, giving a thrilling account of the

RAIL-SECTIONS.

BY FREDERIC A. DELANO, CHICAGO, ILL.

(New York Meeting, February, 1889.)

THE subject of the wear of rails seems to have attracted an unusual amount of interest in the last six months, and in the hope of doing my share to direct opinions in what seems to me the right direction, I will attempt to discuss briefly the several ways in which the shape of the rail-section may affect the life of rails. In a paper presented at the Boston meeting of the Institute (*Trans.*, xvi., 594),

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tragedy. They testified that the party came to a halt in front of the fatal chamber; Schaufuss set up his transit; then he and Owen went to look for marks of a previous survey. Near by was a canvas door which served to turn the air-current through the chamber. Though they lifted the canvas and passed through, the door was not left open. On returning, Schaufuss stood at the transit. Owen was off to the right, and Riley, Snyder and Leiber stood near the transit. Riley held the reel on which the steel tape was wound. Rudge took the end of the tape and started up the steep slope of the breast, his naked lamp on his hat and a safety-lamp in his hand. Two of the witnesses are positive as to the lights being thus carried. When he had proceeded about 20 feet, one witness says he saw the gas ignite from the naked lamp, the flames sweeping down towards the party. All threw themselves on their faces and the flames swept over them. Their lights were out and they immediately began groping their way out. While they were thus occupied, they say, a second explosion took place. They were presently overtaken by Rudge, the fire-boss, his clothes ablaze. One of the men stopped him and tore the burning clothing from his back.

"Relighting a lamp, they started to return in search of their comrades, Schaufuss and Owen, who were missing. They were unable to reach the spot on account of the after-damp and dense smoke from the burning timber. Turner, the foreman, was summoned; and he made a similar attempt, but was driven back. Summoning other assistants, he stationed one man at a door which he was instructed to open for an instant, then close, repeating the operation constantly. In this way a part of the air-current was diverted from other parts of the mine. The smoke being partly driven out of the deadly chamber, they found first Owen, who had tried to run through the flames and became suffocated. Then they discovered Schaufuss, buried by a scale of rock splintered from the roof by the intense heat. Schaufuss, like his companions, had thrown himself on his face, with his hands under him, to save them from the flames. His clothing taking fire, he was not aware that the blaze had swept by. He therefore lay still too long and was suffocated by the after-damp. The explosion was very light, the board brattice not being destroyed. The heat, however, was very intense. This is shown by the fact that, on Monday, Inspector Williams and others found the steel tape where Riley had dropped it, the coils being solidly united by the melting of the solder on which the figures are engraved. Just 24 feet had been uncoiled. The hard wood legs of the transit were burned up, and the instruments totally destroyed.

"Rudge, the fire-boss, asserts that on entering the chamber he took off the naked lamp and put it on the ground at the entrance."

I outlined the principal causes in the various steps of manufacture which might affect detrimentally the service of steel rails, showing that, in my opinion, the result might be algebraically represented by an equation composed of a number of variable factors, and that it was not possible to ensure a good rail by fixing any single one of these. In the present paper, on the other hand, I shall seek to prove that the design of the rail-section, independent of the material used, may make a poorly-wearing rail.

The designing of many rail-sections by engineers who are far from being experts in the metallurgy of steel, has led to the general acceptance of erroneous principles as the rules for such designs. Until very recently, it has been assumed that the more metal there was in the head, to wear away, the longer the rail would wear; and it has been the practice, in adopting heavier sections, to put all the additional weight on top of the rail-head of a very much lighter section. Careful investigation on a number of railroads has shown that of the rails weighing 65 lbs. per yard and over, the greater part have to be removed after comparatively trifling abrasion, solely because they are no longer fit for a smooth track. This is evidence of error in the principles of the design of the section.

It is generally admitted that, as a rule, rail-sections designed since 1879, for rails of 65 lbs. and over, have not given entire satisfaction. There are several reasons which may account for this: first, a constant increase, during this period, of wheel-loads on engines and cars, together with generally higher speeds; secondly, new methods in manufacture, cheapening the cost of production and possibly producing an inferior article; thirdly, mechanical conditions in the rail itself, due to the shape of the rail-section.

1. The first cause is undoubtedly a real one. I think there is evidence that the elastic limit of much of the rail-steel now in use is often actually exceeded by the strains of practice, causing a flow of metal, which is then abraded by the wheel flanges; and the result is a kinking or a battering of the rail, which condemns it for a good track. Granting the existence of this cause, there are two ways of ameliorating the result; one is to increase the bearing-surface of wheels on rails; the other, to raise the elastic limit of the metal, which subject I shall take up later on.

2. With regard to the second cause, it is held by many that the enormous production, low cost, and "improvements" induced by competition have involved inferior quality of product. Many well-informed men are content to rest on the simple assertion that the

old John Brown steel gave remarkable satisfaction, because it was made of a good iron and did not contain more than .08 per cent. of phosphorus, any analysis showing more being grossly inaccurate. If they are satisfied that this is the solution of the problem of making a really good rail, why do they not try their .08 per cent. of phosphorus? Low phosphorus steel can be made to-day with more certainty and at less cost than ever before. The only question is, whether the elimination of phosphorus is going to be worth what it costs. The fact that as many Bessemer charges are now blown in one hour as were formerly blown in a day, is pointed at as evidence that the same care cannot be given to the operation; but I think men really conversant with the turning-out of a large product will maintain that almost absolute uniformity in the iron is necessary to keep things running with the clock-work regularity required for such a rate of production. Indeed, large product and general smoothness in mill-operation are, to my mind, an indication of a uniform product, so far as metal in the rail itself is concerned, and seem to me, therefore, not objectionable on that score. With the rapid rolling, however, particularly the rapid blooming or breaking-down of large ingots, I find fault, as I have explained at some length in my former paper, already cited.

3. That mechanical conditions in the rail itself, due to the general design of the rail-section, can have any important effect on the service-value of rails has not been generally admitted, so I wish to lay particular stress upon this cause and attract to it the recognition it deserves. Speaking in general terms, that form of rail is, in my opinion, faulty, which has a deep massive head, a wide, thin base and a thin web, connected to the head and the base by small fillets only. The objection to this form and the advantages which might be obtained by different proportions, may be concisely enumerated as follows:

a. Such a form necessitates rolling at a higher temperature, and, what is of chief importance, finishing at a higher temperature than permits the securing of a well-forged, compact rail.

b. This type of section probably encourages the use of a higher percentage of manganese to reduce the number of second-quality rails; and this production of second-quality rails, as well as the additional manganese, figure in the cost of the rails to the consumer.

c. Rails with heads out of proportion to the rest of the section require a great deal more cold-straightening, because it is impossible

to know how much bending or cambering such a rail needs in its hot state in order that it shall cool straight. This is certainly a very important consideration.

d. The same sort of disproportion must produce corresponding disproportions in the rate of cooling, causing internal strains which are only partly dispersed or effaced by the small connecting fillets.

e. A section having these internal strains exaggerates the effect of any impurity which may chance to have segregated in the ingot, while a section not liable to these strains would safely admit of a generally higher carbon steel as well as a considerable latitude in composition.

f. It is not economical to provide for an abrasion of some $\frac{1}{8}$ of an inch, or even more, from the top of the rail, if it is found that rails generally fail long before this amount has been abraded. Instead of providing more height to wear away, we can do better by providing more breadth of bearing-surface, and metal of a higher resistance within the elastic limit.

g. Inasmuch as the funnel-shaped cavity, dispersed blowholes and possible impurities exist in the ingot in the axis of its length, it follows that these defects will exist (especially in rails made from the upper part of the ingot) on an axial line which passes through the rail-section. It follows that if the rail-section has 50 to 55 per cent. of its metal in the head, the poorest metal is enclosed in the head, where it receives the least amount of forging; whereas, if the amount of metal in the head does not exceed 40 to 45 per cent., this poor metal occurs in the thoroughly-worked web portions.

The foregoing objections seem to be so formidable and convincing that I am glad to see a general tendency among all the later investigators to come to the same conclusions. It seems to me that rail-sections should be made to certain definite proportions, regardless of the weight per yard. It is often said that the larger sections will admit the use of a higher carbon-steel; but this assertion requires the proviso that the heavier section shall be similarly proportioned. One seventy-four-pound section which I have in mind could not be safely made of as high a carbon steel as the sixty-two-pound section from which it has been schemed.

While I am deeply interested in the recently published researches of Mr. D. J. Whittemore, Chief Engineer of the Chicago, Milwaukee and St. Paul Railroad, and agree entirely with him in his conclusions that wheels should be cylindrical and the bearing-surface should be increased, it seems to me that a rail-head of twelve inches

radius is flat enough. Since there is a lateral play of fully half an inch for the average car-wheel on an average track (not to mention a four foot nine inch gauge) both the wheel treads and the rail-heads must wear to a curve. On the other hand, if the rail-head begins with a slight top-curve, it would take no longer than with the flat-topped rail to assume the curve of the average wheel. This is a subject, however, which deserves more attention than I am able to give it in the present paper. Mr. Whittemore has ably argued the need of more bearing-surface, which, after all, can only be obtained by widening the rail-head; and I am glad to quote him here, as authority for the need of wider heads for our rails. The reasons for making the head shallow, as well as wide, have been recited. For the same reasons the web and flanges should be heavy, with liberal connecting fillets.

The problem is to increase the life of rails without greatly enhancing the cost. It might be possible to make a sixty-pound rail of steel worth $75 per ton give as good service as an eighty-five-pound rail made of $30 steel; but the discovery would not prove remunerative.

With regard to internal fillets and flat fishing-angles, I have held the extreme view, that the fillets should be liberal and the angles of enough slope (certainly not less than 15 degrees) to allow really colder rolling than is anywhere the practice in this country.

The specifications which I append as a conclusion to these brief remarks may be of interest to some, as being different in several particulars from any other rail specifications. They were drawn up originally fifteen months ago, and have not been changed in any important particular since that time.

Proposed Specifications for Steel Rails.

1. The steel used in the rails shall be made by the Bessemer or by the Open-Hearth process and shall not contain more than $\frac{11}{100}$ths of one per cent. of phosphorus, as determined by an analysis made from a boring into the center of metal of the rail-section of any rail.

2. The ingots shall be poured with care, and the moulds filled at one pouring, without undue spattering of the sides. No sand or water shall be thrown upon the ingots after pouring; but, on the contrary, coke or charcoal-dust shall be used. In no case shall rails made from a chilled heat or a bled ingot be accepted as first-quality rails.

3. Cold moulds, or moulds in such a shape as to require more than a hand-sledge to free them from the ingot, shall not be used.

4. It is preferable that ingots, when taken from the pit, should be kept erect and so heated; but if this is not the practice of the rolling mill company the ingot shall not be laid on its side until it is certain that the central part has solidified.

5. Blooming-rolls of forty inches from center to center shall not be driven at a speed exceeding thirty revolutions per minute, and larger or smaller rolls shall not be driven at a speed exceeding this in linear velocity. (The linear velocity of an ingot going through the first or thirteen-inch pass of a pair of forty-inch rolls, driven at a speed of thirty revolutions per minute would be about 205 feet per minute.)

6. No metal shall be cut from that end of the bloom which corresponds to the "butt" end of the ingot.

7. From that part of the bloom which corresponds to the top part of the ingot an amount shall be cut equalling in pounds the area of the mean cross-section of the ingot in inches, or as much more as is necessary to secure a sound pipe-free bloom.

8. The number of the charge shall be stamped distinctly on each rail and the place and date of manufacture rolled on.

9. The section of the rail rolled shall conform to the template, and a variation of more than a scant thirty-second of an inch over or under the template height shall not be allowed, and the rail must be made close enough to template to fit the splice properly.

10. The weight of the rails per yard shall be kept as close to the standard weight per yard as can be after complying with the conditions of clause 9.

11. The rails shall be drilled for the bolts of the splice-bar, or, if necessary, notched in the flange, in accordance with the templates and diagrams furnished by the railroad company.

12. The rails shall be cut at right-angles to their length, and be at 60° to 75° F., within one-quarter of an inch of the standard length of thirty feet. No shorter length rails shall be accepted by the railroad company, except under special provisions of this contract for such acceptance.

13. Rails shall be perfectly straightened in both surface and line, without any twists, waves, or kinks. Particular attention shall be given to having the rails free from "camber," in which the ends droop, and also free from any flaws, cracks, or excessive roughness.

14. The rails shall be finished at such a temperature that it will not be necessary to allow more than four inches for contraction on every thirty-foot rail sawed immediately after coming from the finishing-pass in the rolls.

15. One rail-end, which must correspond to what has been the ingot top, shall be tested for each heat or charge by a drop-weight of one gross ton, falling ten feet on the rail-head, midway between the supports, three feet apart. (The bearings are to be of iron or steel, fastened firmly to oak framing constructed of oak timbers bolted together into a solid floor, and this supported on masonry foundations not less that four feet deep. The guides are to be smooth, straight, and parallel.) Should one rail-end be broken by the above test, two similar rail-ends must succeed in enduring the shock, or else cause the rejection of all rails in that same heat. It is expressly desired that the rail-steel shall contain as high a percentage of carbon as the maker is willing to put in and still produce a rail which shall endure this drop-test.

16. The representative of the railroad company shall at all times be allowed, on request, to see the *bona fide* analyses and carbon-determinations, as made by the rolling mill company, of the steel used in making these rails.

HOLLOW IRON PIG-PATTERNS.

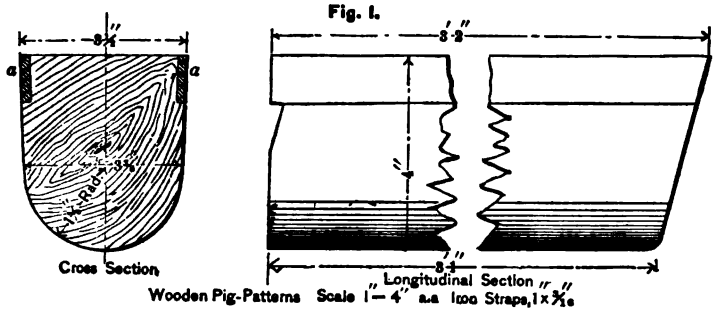
BY B. F. FACKENTHAL, JR., RIEGELSVILLE, PA.

(New York Meeting, February, 1889.)

FOR the past year we have had in use at the Durham furnace a set of hollow pig-patterns made of iron, which have given such satisfactory results that I think a description of them would be of interest to the members of the Institute.

Before the introduction of these iron patterns, we used the ordinary wooden patterns, made in the usual way with straps of iron along the sides to protect the corners from wearing off. The size and shape of these wooden patterns are shown in Fig. 1.

Fig. 1.

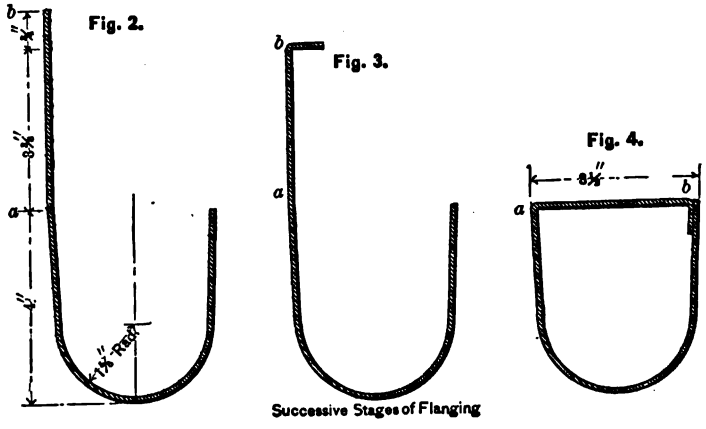


When the patterns were new, the pigs weighed about 130 pounds each, and were uniform in size, and the moulding was clean. But constant use soon wore the patterns away; and at the end of a few months, the pigs had become, in consequence, so much lighter that the moulding, daily, of several additional beds was necessary to accommodate the same quantity of iron. As the patterns wear away, the moulding becomes ragged and the pigs rough. Moreover, wooden patterns are often destroyed by fire. I have known an entire set to be destroyed by a break-out of iron.

The iron pattern which I have designed is free from all the objections to the wooden pattern, being not only more durable, but, in the long run, far cheaper.

The iron pattern having the same shape and size as the wooden one, is made of the best flange-iron, of No. 13 gauge. After the sheets have been cut to the proper size, three heats are required for the flanging. At the first heat, each piece is stamped in a cast-iron form, which gives the proper shape to the bottom part of the pat-

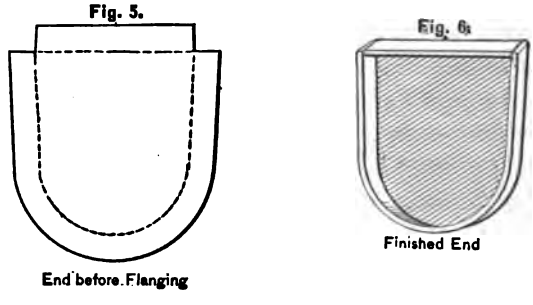
tern, as shown in Fig. 2. At the second heat, it is flanged at *b*, as shown in Fig. 3. At the third heat, it is flanged at *a*, giving the pattern its final shape, as shown in Fig. 4.



The flanging at *a* and *b* is done on a square mandrel.

Care should be taken to have the corners, *a* and *b*, left full and perfectly square.

It now remains only to rivet the pattern together, and put the heads or ends in. The ends are also made of No. 13 flange-iron and are stamped in a cast-iron form or die by means of a screw-punch, the iron being cut to the proper shape before stamping, as shown in Fig. 5.



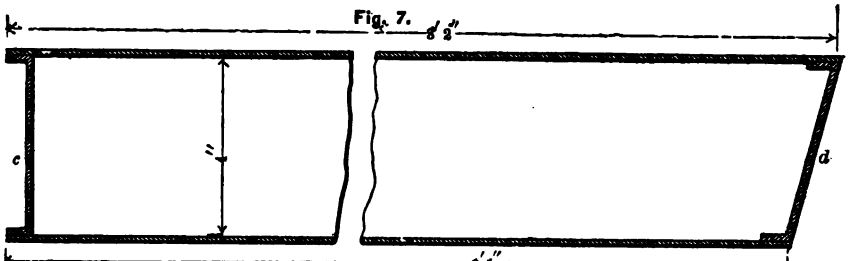
These ends can be made very quickly, only a few seconds being required for the stamping. The finished end is shown in Fig. 6 and at *c* and *d*, Fig. 7.

The end farthest from the sow, and marked *d* in Fig. 7, is, of course, put in first, the flange going inside of the body of the pattern as shown. The end next to the sow is then put in with the flanged part to the outside, as shown at *c* in Fig. 7. This forms the

handhold for drawing the pattern out of the sand. The rivets are placed about 1½ inches apart, and are all countersunk on the outside, the holes being drilled after the pattern is flanged.

A finished pattern of this kind weighs no more than a new wooden pattern after it has become water-soaked. Thinner iron could be used; but I have thought it best to use No. 13, and thus

Fig. 7.



Longitudinal section of Iron Pig-Patterns

make a strong pattern, capable of withstanding the rough usage to which pig-bed patterns are subject. The patterns which we have had in use during the past year show but little wear. They are smooth, though the slight oxidation to which they are subject prevents them from getting very bright. While we have had no trouble with water leaking through the seams, the joints remaining tight, I have thought that it would be an improvement to braze the joints. A very neat pattern could also be made by casting it of bronze; but this would be more costly, and I do not see that it would be any better. The set of patterns in use at Durham cost about $1.75 each, of which seventy-five cents is the cost of the iron. They can, doubtless, be made more cheaply when the men become more skilful in the work.

NOTE ON THE KOEPE SYSTEM OF WINDING FROM SHAFTS.

BY JOHN H. HARDEN, PHŒNIXVILLE, PA.

(New York Meeting, February, 1889.)

THE Koepe system of winding from shafts is the invention of Mr. Frederick Koepe, Manager of the Hanover Coal Mine, Westphalia, one of the collieries worked by Krupp, the well-known German ironmaster. The first trial of the system was made at the Hanover

shaft, and proved so successful that several engines belonging to the same owner were reconstructed on the new plan. From an examination by inspectors it was demonstrated to the satisfaction of the German government that the system was not only in every respect more economical than the ordinary method of hoisting, but had also elements of greater safety to recommend its universal adoption. The plan has since been adopted at several mines in the same district and others in France, and still later at the Bestwood colliery, near Nottingham, England. The principal and simplest form of the Koepe system can be applied to existing hoisting-machines at little cost, by attaching and suspending an old hoisting-rope from the under-side of the cage at the top of the shaft and connecting the opposite end to the under-side of the cage at the bottom of the shaft. This balance-rope may hang loosely in the shaft, or it may pass round a tightening-sheave at the bottom. In this way all the parts of the hoisting-apparatus are perfectly balanced throughout the lift, which is not the case by any other system of adjustment. There are several modifications of the system; but the principle is the same in all cases.

The advantages claimed for the Koepe system are as follows:

1. Balancing the hoisting-ropes, thus reducing the size of engine, consumption of fuel and cost of plant for a given duty.
2. The same engines will raise a given load from any depth in the shaft.
3. One rope only is necessary instead of two.
4. Safety from over-winding.
5. Less wear and tear of plant.

The benefit will be more readily appreciated by taking examples of engines fitted with parallel drum, in comparison with the Koepe engines with winding-pulley and counterbalancing tail-rope. This, according to the greater or less depth of shaft, varies from 35 to 64 per cent. of the power required to lift the unbalanced load. Wherever parallel drums are used it is possible to increase the duty of the engine by the addition of a balance-rope.

Quite recently a plan has been adopted at a Pennsylvania colliery, said to be a new invention, and called the "Poore" system, which is believed to be nothing more than a modification of Mr. Frederick Koepe's invention.

It is not necessary to enter into further particulars. The system will be found fully described and illustrated in the *Transactions of the Chesterfield and Derbyshire Institute of Mining, Civil and*

Mechanical Engineers, vols. xi. and xii., 1882–83, from which I have gathered my information.

Discussion.

Herbert W. Hughes, Dudley, England. (Communication to the Secretary). Mr. Harden has, I think, scarcely made plain the principle of Koepe's system of winding. Briefly stated, it consists in substituting for the ordinary cylindrical drum a grooved pulley round which the rope makes rather more than half a turn, and thence passes over the pit-head pulleys and down the two divisions of the shaft. The balance-rope beneath the cages is by no means a peculiarity of the system, as it has been applied for a long time to numerous winding-engines in England, where ordinary cylindrical drums are used. Experiments on the Koepe system have shown that with the rope passing only one-half turn round the driving-pulley, the co-efficient of adhesion between steel rope and wood rim is in practice 30 per cent. which admits of an excess of 105 cwt. being placed on the present ascending load at the Hanover colliery before any slip can occur. That no slip actually results in practice (under the *usual* working conditions) is shown by the fact that at Bestwood colliery the winding takes place at the upcast shaft which is cased in, and the cages are entirely out of sight of the engineer, who has to rely entirely on the indicator, and under these circumstances has no difficulty in landing the load. It is, however, evident that when the cages reach the landing-places and rest on the stops (if any are used) the weight is removed from the rope and sufficient adhesive power may not exist on the rim of the motive-pulley to enable the loads to be restarted. This can be guarded against by dispensing with stops altogether (as is done at the Sneyd colliery), or by continuing the rope past the cages by means of cross-heads above and below each cage connected together by side pieces passing outside; the bridle chains are hung from the top cross-head and when the cage rests on the stops the weight of the winding- and tail-ropes still remains on the motive-pulley. This is the arrangement used at Bestwood.

The single winding-rope at the Hanover colliery has been found to last more than twice as long as the two ropes formerly adopted.

The chief advantage of the system, apart from the perfect equalization of the load, which can also be obtained in any engine with ordinary cylindrical drums, consists in doing away with the drum which in many instances weighs 60 tons and has of course to be set

in rapid motion and stopped in a short space of time, causing a large waste of energy. The merits and demerits of the system are fully examined in an elaborate inquiry by M. L. Trasenster, *Revue Universelle des Mines*, vol. v., 1879, p. 85, and later particulars are given of the results obtained at the Hanover colliery in a report by Messrs. Mahlet, de Gournay and Suisse on "Mining Appliances in Westphalia," *Bulletin de la Société de l'Industrie Minérale de St. Etienne*, 1887, vol. i., pp. 65, 389. The Koepe system of winding has been adopted at Oberhausen and Westhausen in Westphalia, Stassfurth, in Upper Silesia, and Bestwood and Sneyd collieries in England, but has been abandoned at Oberhausen and Westhausen for reasons one is unable to ascertain, possibly that breakage of one rope would cause the stoppage of both sides of the pit. After seven years' successful working, Koepe's system has been lately abandoned at Bestwood for two reasons: the management do not consider it safe, and slipping of the rope takes place every time it is oiled; this slipping commences immediately oil is applied, and after a time ceases altogether, to re-start; however, at the next oiling. This action is very objectionable at Bestwood, for, as before stated, the engineman has to rely entirely on the indicator for landing the cages, as he cannot see them when they reach the surface owing to the top of the shaft being cased in.

Against these abandonments we have the fact that no accident has occurred at the Hanover pit since the installation was put down in 1877, indeed the life of winding-ropes is increased as before stated. The system is also giving every satisfaction at the Sneyd colliery in North Staffordshire.

NOTES ON THE ROSARIO MINE AT SAN JUANCITO, HONDURAS, C. A.

BY THOMAS H. LEGGETT, FAIRPLAY, COLORADO.

(Buffalo Meeting, October, 1888.)

THE conditions surrounding this mine are, perhaps, not sufficiently rare or significant to warrant special attention. A certain interest, however, attaches to the locality through the recent discovery of fossil cycads made there by Mr. Charles M. Rolker, of New York, who has repeatedly visited this district, fixing the geological horizon

of the vicinity as Upper Triassic. Moreover, the Rosario mine, being by far the largest producer of Central America (the value of the bullion output in 1887 being about $610,000 in gold and silver), may, on that account, merit notice in the *Transactions* of the Institute.

Undoubtedly, many of the mines of Honduras have been greatly over-rated. The scrutiny of investors seems to decrease, and their desire to buy to augment, in direct ratio to the square of their distance from the foreign mining-country. It is also true that the country has had its full quota of mines ruined through mismanagement; nevertheless, it has to-day mines that are producing well, and the following short description of one of them may prove not wholly devoid of interest.

Location and Topography.

To reach the interior of Honduras from the Pacific side, one disembarks from the coast-steamers of the Pacific Mail line at the port of Amapala, which is situated on Tiger Island, in the Gulf of Fonseca.

The town lies close to the water's edge and presents the characteristic Spanish-American appearance of low, white-plastered house-fronts, over-topped by the shining belfry-towers of the church on the Plaza, while immediately in the rear of the town a cone-shaped mountain rises to the height of nearly a thousand feet.

This, like many of the surrounding island-mountains, was at one time, to judge from the lava which strews the shores of Tiger Island, an active volcano.

To reach the mainland from Amapala, a small boat conveys one across the north end of the gulf, and up narrow creeks that are often completely arched over by the dense tropical foliage, to the pueblo (hamlet) of San Lorenzo, twenty miles distant from Tiger Island.

Thence a wagon-road, built for ox-cart traffic, and over which travel on mule-back is comparatively comfortable, leads northerly ninety-six miles to the city of Tegucigalpa, the capital of the country. The first forty miles of road lie through flat but slowly rising lands, flanked on the east and west by hills, spurs from the main range further north, and watered by a river which, in the dry season, is 50 yards wide, but nearer 200 yards in width in the rainy months. This river is called by different names in different localities, as the Perspire, Choluteca, Rio Grande, etc. The remaining distance is

over a rough mountain range of eruptive origin, with a general northwesterly and southeasterly trend, showing basalt, trachyte, porphyries and felsites in abundance, and attaining altitudes of 3500 to 4500 feet above sea-level.

The Cerro de Hule (possibly so called from the absence of caoutchouc trees in its vicinity) lies about 17 miles southwest of Tegucigalpa, and is 4050 feet high where the road crosses the mesa on its top. This is the greatest elevation attained between the capital and the coast.

From the town of Sabana Grande, one travels for six leagues north-northeast to the "Cerro," across table-lands over 3000 feet above sea-level, and covered with a heavy growth of pine timber. In the vicinity of Sabana Grande, the first marls and limestones are encountered, while on the northerly flank of the Cerro de Hule, well down towards the Valley of Tegucigalpa, marls and cretaceous tufas are met with. Through these latter an excellent road has been cut, leading up the valley for 15 miles to the city of Tegucigalpa, which lies in latitude 14° 15′ North, longitude 87° 10′ West, and 2465 to 2500 feet above sea-level.

The town of San Juancito is situated 19 miles northeasterly from the capital, in a gulch on the east flank of the Plazuelas mountains, at the level of the pine-belt, or an altitude of 3050 feet above sea-level, and at the junction of the San Juan and Escobales mountain-streams. Its climate is even, the temperature rarely rising higher than 85° F., or falling lower than 50° F. It is disagreeable, however, during the rainy season, which usually lasts from May into October.

The topography of the country surrounding San Juancito is very rugged and mountainous. The Plazuelas-Montañita range, running a little east of north, divides the Valley of Tegucigalpa, at a mean altitude of 2500 feet, from the Cantarranas valley, which lies east of the range at 1600 to 1800 feet above sea-level, while the Crucero and Chanchera mountain peaks of that range attain altitudes of 6800 and 7000 feet.

The range above the pine-belt, which seldom reaches as high as 4500 feet altitude, is covered with a heavy growth of oak and other hard-wood trees, together with some of the varieties of the tropical soft woods. Most of the hard woods afford an excellent mine-timber. Very cool and refreshing, after the hot and dusty travel of the valleys, is a ride through these forests of big trees, their dark-green foliage relieved by thickly studded brilliant-hued parasites, and

their trunks almost hidden by clinging vines, and the immense fern-trees growing at their roots.

Ten to twelve miles easterly, across the Cantarranas valley, runs the Chili range of mountains, parallel to the first-mentioned range and reaching nearly equal altitudes, while coming into the valley from the west through a break in the Plazuelas range the Rio Grande flows southerly through it, and the succeeding valley of San Francisco, towards the Gulf of Fonseca. These valleys are broad and flat, dotted with irrigated ranches, growing corn and sugar cane, and affording good grazing lands for cattle.

Across the Cantarranas valley, about 18 miles northerly from San Juancito, near the town of Talanga, runs a low divide, which separates the valley of the Ulna river to the north, from that of the Rio Grande to the south; or the tributaries of the Atlantic from those of the Pacific.

Geological Features in the Vicinity of the Mine.

The geology of this section of the country is well shown in the map and section (Figs. 1 and 2) of the San Juancito mining-district, given me by Mr. C. M. Rolker. In the sectional map, Mr. Rolker represents the successive sedimentary strata as follows, their respective thicknesses being shown.

Commencing at the top there is found:

Limestone (gray), Bermudez Portillo and Valley Pass.

Fine red conglomerate (shows effects of proximity to heat).

Red aluminous sandstone, more or less micaceous, changing locally to slate-shales.

Fine red conglomerate (the red color shows proximity of heat).

Limestone (Peten, possibly the same as at Cantarranas).

As already noted, the Chili range lies to the east of the Cantarranas valley and the Plazuelas-Montañita range to the west. The breaking through of these ranges was perhaps accompanied by a sinking of the valley, the limestone of which was tilted on the east flank of the western foot-hills. It may be identical with that of the Peten mine; but this cannot be asserted at present.

The fossil cycads were found in the shales and sandstones at the point marked on the map, Fig. 1. They are described, in the *American Journal of Science*, for November, 1888, p. 342, by Professor J. S. Newberry, who declares these strata to be of Upper Triassic age.

The overlying fine red conglomerate and Portillo and Valley

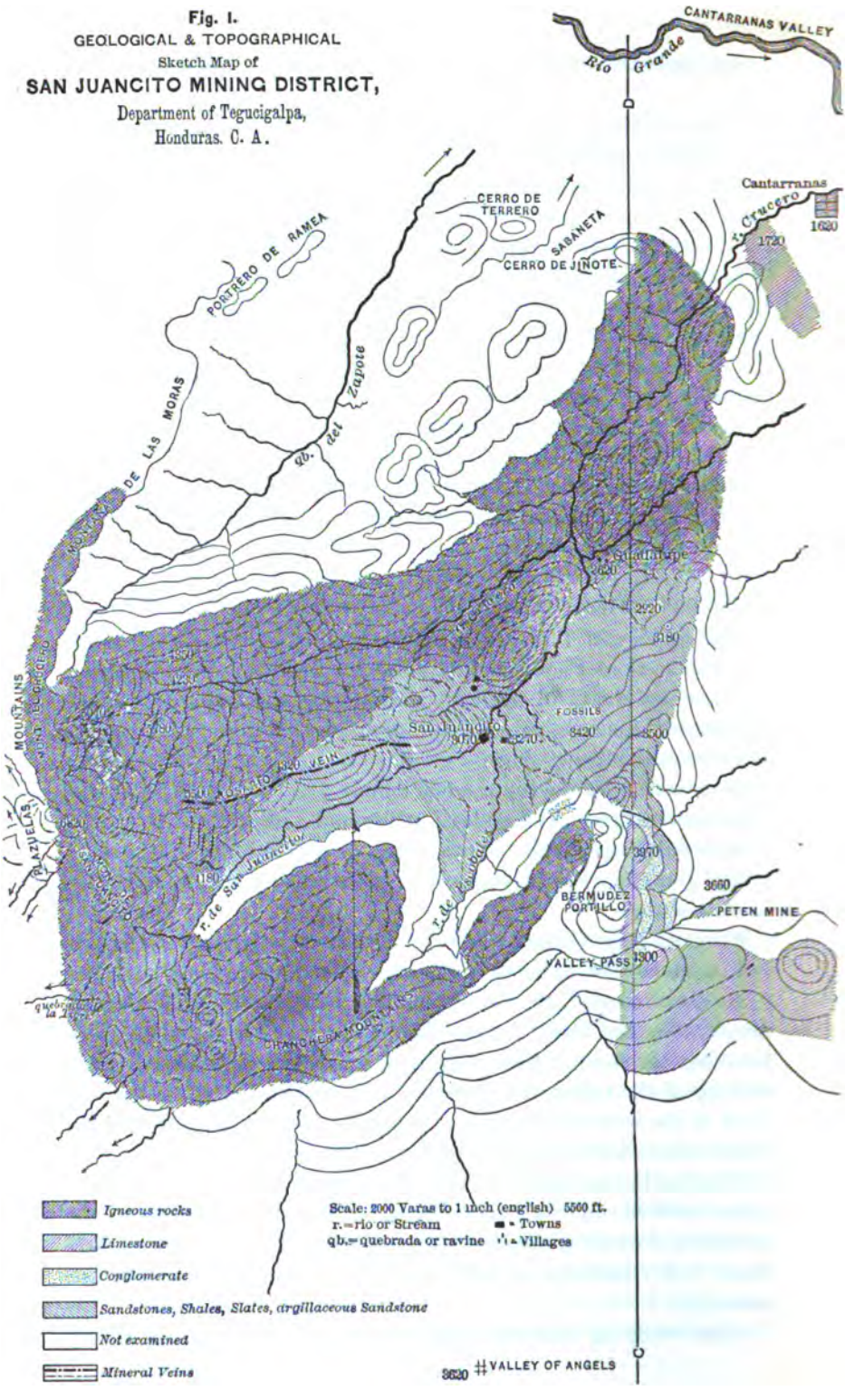


Fig. I.
GEOLOGICAL & TOPOGRAPHICAL
Sketch Map of
SAN JUANCITO MINING DISTRICT,
Department of Tegucigalpa,
Honduras. C. A.
CANTARRANAS VALLEY
Río Grande
Cantarranas
Crucero
1620
1720
CERRO DE TERRERO
SABANETA
CERRO DE JINOTE
POTRERO DE RAMEA
qb. del Zapote
MONTAÑA DE LAS MORAS
MOUNTAINS
GUAZUELAS
Guadalupe
2920
3180
FOSSILS
San Juancito
3270
3420
3500
3970
3660
PETEN MINE
BERMUDEZ
PORTILLO
VALLEY PASS
3300
r. de San Juancito
r. de Bermudez
CHANCHERA MOUNTAINS
Igneous rocks
Limestone
Conglomerate
Sandstones, Shales, Slates, argillaceous Sandstone
Not examined
Mineral Veins
Scale: 2000 Varas to 1 inch (english) 5560 ft.
r.=rio or Stream
Towns
qb.=quebrada or ravine
Villages
3620
VALLEY OF ANGELS

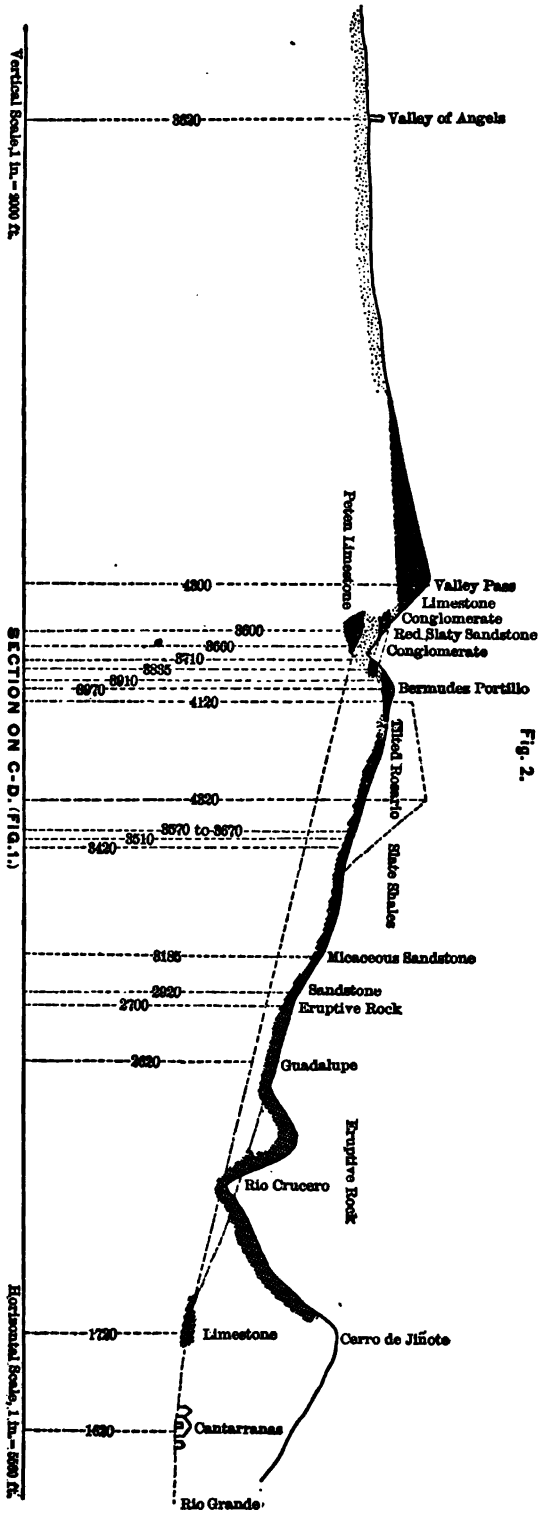

Vertical Scale, 1 In. = 3000 ft.
SECTION ON C-D. (FIG. 1.)
Horizontal Scale, 1 In. = 3000 ft.
3620
Valley of Angels
4300
Peten Limestone
Valley Pass Limestone
3600
Conglomerate
Red Slaty Sandstone
3600
3710
Conglomerate
3335
3910
3970
Bermudez Portillo
4120
Tilcal Rosario
4320
3970 to 3670
3510
3420
Slate Shales
3185
Micaceous Sandstone
3020
Sandstone
2700
Eruptive Rock
2620
Guadalupe
Eruptive Rock
Rio Crucero
1720
Limestone
Cerro de Jilote
1420
Cantarranas
Rio Grande

Fig. 2.

limestone belong probably to the Jurassic period. Unfortunately, no fossils were discovered in them.

Southwest of San Juancito, in the Tegucigalpa valley, marls and sandstones occur, and of this Mr. Rolker says: "When I go south from the Valley of the Angels toward Santa Lucia (a town about three leagues west of Tegucigalpa) up the grade, I find the Triassic sandstones again raised and tilted to the south, but 800 feet higher, shading out towards Santa Lucia and alternating with marls towards Travesias and Tegucigalpa, where are finally the true marls. That would make San Juancito shales and slates and the Tegucigalpa and Santa Lucia marls originally one marine stratum. Then the Plazuelas (San Juancito, Crucero and Moras) mountains with the parallel Montañita mountains broke through, the two ranges connecting back (west) of the Valley of the Angels by the Chanchera ridge. This tipped the sandstones and marls south of the Valley Santa-Lucia summit southwesterly, and southerly and north of the Portillo northeasterly. In other words, what is marl at Tegucigalpa and Santa Lucia, is shales and slates at San Juancito, and below these lies the conglomerate (second fine red conglomerate) showing between Bermudez Portillo and Valley Pass. The upper, Jurassic, conglomerate and limestone are only visible on these two 'Portillos' near the Valley of the Angels and have elsewhere been scored off."

The map, Fig. 1, shows the Rosario vein as lying partially in sedimentary and partially in eruptive rock, the latter forming the central core of the Plazuelas mountain range. Its intrusion uplifted and contorted the overlying bedded rocks, tilting the present edges of the slate-shales on the eastern flank of San Juancito mountain, over 1000 feet above their normal level, under an angle of from 30° to 60° to the north. Their strike is approximately east and west.

The intrusive rock is of a compact texture, very rarely showing the imbedded feldspar crystals, and of a light grayish-green color. From four specimens sent to Mr. F. J. H. Merrill, of the School of Mines, Columbia College, this rock was determined by him to be rhyolite, and composed of crystals of oligoclase in a felsitic matrix. He describes no accessory constituents whatsoever; and doubtless the rock proved as barren of these to the microscope as it does to the ordinary lens.

In the vicinity of the vein there is no such definite contact between the slate-shales and intrusive rhyolite as exists to a much greater extent in other parts of the mountain. Here, the two inter-

lock in a sort of fringed edge, the rhyolite penetrating between the strata and raising them. The stringers of slate-shales pinch out, however, as depth into the mountain is attained.

One principal strip of slaty-shale, argillaceous in character, shows 120 feet wide to the north of the vein at the 400-feet level. After crossing the vein, it shows 90 feet in width south of the latter, and at the zero-level. At a point 350 feet in from (west of) the mouth of this level, it pinches and disappears entirely. Figs. 6 and 7 show this and the interlamination of the slate-shales and rhyolite. This argillaceous shale frequently shades into slate on and near the surface, showing all its characteristics except that of cleavage; but with depth it becomes a typical black shale, somewhat baked through the influence of the near eruptive rock.

There are evidences of an immense amount of erosion in this vicinity and over the vein. Huge boulders, 60 and 80 feet in diameter, lie in the ravines 1000 feet and more below the mine. Further, the overlying conglomerate and limestone showing above these slate-shales on the ridge to the east of San Juancito (see Fig. 1) have been entirely scored off and do not show anywhere on the mountain range west of the Rio Escobales.

The regular course of the vein through the sedimentary and eruptive rock, as may be demonstrated by tracing the fissure through the interlamination of the two rocks, shows its origin to have been somewhat later than the upheaval of the latter.

Vein-Characteristics.

The Rosario vein is a quartz lode, and may be classed as a "diaclase-" or "fissure-fault,"* cutting through sedimentary and eruptive rock alike, and having been traced for over 6000 feet along the surface and claims located thereon, as shown in Fig. 8. It strikes north 74° 30′ east for more than three-fourths of its length, and until nearing the Matraca claim, where it swings 8° to 10° nearer east. While there are numerous local curves and bends in the vein, its average course is very constant. It dips to the north in places as steeply as 70° from the horizontal; in others, 10° and 15° less. Its average dip is 63°; and this it holds with great regularity as distance into the mountain, or depth from the surface, is attained. The vein pinches and widens in the manner common to such lodes. At some places it is only 2 feet wide, at others (usually in ore-bodies)

* "Structural Relations of Ore-Deposits," by S. F. Emmons, *Trans.*, xvi., 817.

16 feet between walls. Its mean width can be safely placed at 4½ feet.

The wall-rocks are in places the argillaceous shales, shading into slate as already described. Near the surface, in all the levels, the vein is sometimes entirely in shales, sometimes on the contact-lines of slate-shales and intrusive rhyolite. In all the levels but the lowermost (650-feet level), it finally leaves the former and lies wholly in the eruptive rock.

The walls themselves are well-defined, but seldom equally hard and firm. The foot-wall usually, though sometimes both, is much decomposed and broken in the wider stopes, and a clay parting often runs a foot or two inside of this wall parallel to the vein, necessitating close stulling till the stope can be filled with waste.

The vein-material is quartz, pure and simple; and this shows in 99 per cent. of the vein-exposures throughout the mine. In one or two places only, in the 300- and 400-feet levels and within 150 yards of the mouths of the same, does this give out, leaving but a line of soft talcose clays to mark the vein. This occurs in the 300-feet level for a distance of 70 feet, just on the line of contact of the two "formations." In places also, thin strips of shale enter the vein and pinch out in it, as in the 400-feet level, Fig. 6.

The vein presents most of the characteristics of a "true fissure" lode, in some places giving beautiful instances of banded structure. Its chief peculiarity is a tendency to split into two distinct veins in the more barren ground, and unite into one consolidated vein where the ore-bodies occur. Here its width is oftentimes further increased by the joining of feeders, that usually enter from the hanging-wall. This feature is illustrated by Figs. 3, 4 and 5, a plan, elevation and section of a portion of the 100-, 150- and 200-feet levels, showing the two main veins and the feeders extending into the hanging-wall.

Another important feature of the vein is a combined split and throw, or sharp bend to the south, that occur, more or less marked, in all the levels, a few hundred feet from their mouths.

Where the bend occurs, the foot-wall vein is generally the more prominent and mineralized of the two, the hanging-wall vein being, in some cases, almost obliterated. In the zero and 400-feet levels, where this is most marked, the foot-wall vein finally pinches and becomes barren, necessitating cross-cutting to the hanging-vein, which, in each case, was found to be nearly two feet wide where cut, though indiscernible in the hanging-wall at point of throw of

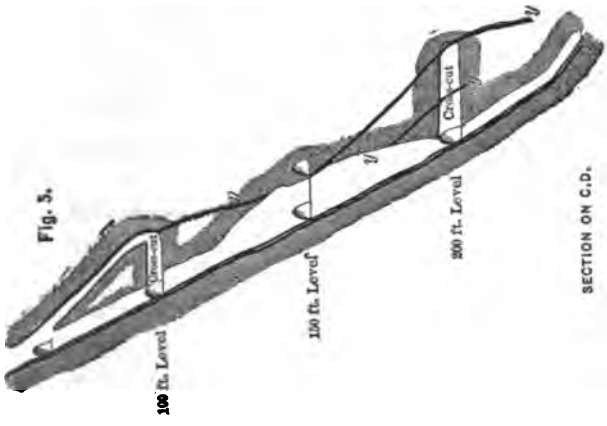


Fig. 3.
100 ft. Level
150 ft. Level
200 ft. Level
SECTION ON C.D.

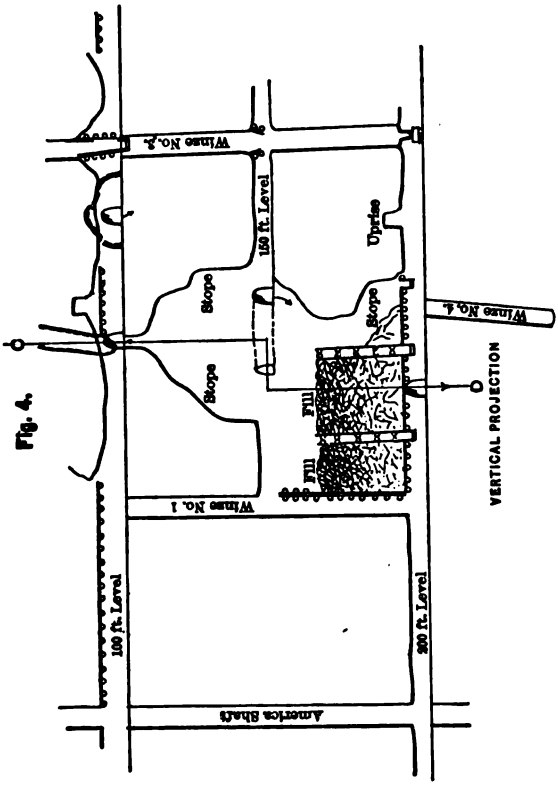
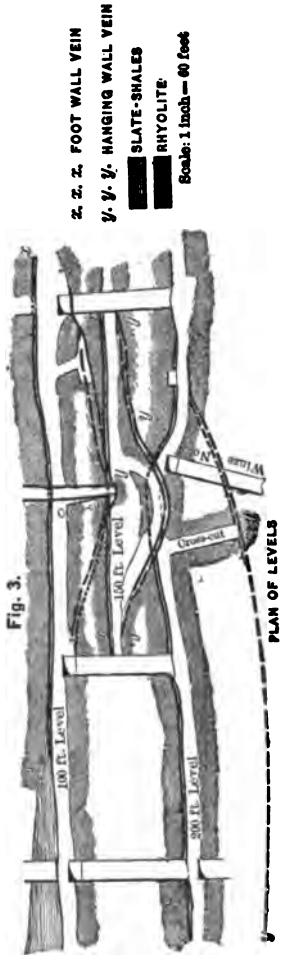


Fig. 4.
Winze No. 2.
150 ft. Level
Dyke
Slope
Slope
Slope
Winze No. 4.
Fill
Fill
Winze No. 1
100 ft. Level
200 ft. Level
America Shaft
VERTICAL PROJECTION


z. z. z. FOOT WALL VEIN
y. y. y. HANGING WALL VEIN
SLATE-SHALES
RHYOLITE
Scale: 1 inch = 60 feet
Fig. 3.
100 ft. Level
150 ft. Level
200 ft. Level
Cross-cut
Winze No. 2
PLAN OF LEVELS

vein. The throw in the intermediate levels is not so sharp; and in these the hanging-wall vein, though bent from its course, continues the stronger and more mineralized. Complete plans and sections would show this more thoroughly, and would be presented herewith did not the Rosario Company object to full copies of its mine-maps being published.

The ore extracted last year was chiefly from the oxidized zone, and was a thoroughly decomposed sulphuret-ore, carrying much native silver and chloride of silver, with free gold and frequent streaks of argentite and other rich silver sulphides. The lower levels show the unchanged sulphides of iron, copper, lead and zinc. The gangue is quartz, carrying in the ore-bodies occasional clay streaks, heavily stained with the hydrated oxides of iron and manganese. Other accompanying minerals, found less frequently, are polybasite, embolite, pyromorphite, wulfenite, cerussite, malachite, azurite, limonite, manganite and pyrolusite.

The ore is treated by raw amalgamation, somewhat to the detriment of quicksilver on account of the manganese present. Frue vanners have recently been added to concentrate the undecomposed sulphurets in the tailings; and the concentrates will be subjected to a chloridizing roasting and amalgamated.

The general trend of the ore-bodies in the mine, as thus far developed, is into the mountain and obliquely downwards, or with a westerly pitch of about 40° from the strike.

This vein is undoubtedly the principal lode of the district; and the fact that it at times follows the contact of eruptive and bedded rock does not in the least prove it to partake of the nature of a contact-vein, as developments have shown it to be equally strong in either rock, and—in proportion to the exploration-work done—almost equally mineralized.

The Mine.

The vein cuts the mountain nearly at right angles to the plane of its eastern slope, and is opened by levels driven in from the surface at vertical distances apart of 100 feet. There are seven of these main levels, varying in length from 600 to 1500 feet (the latter the length of the 200-feet level); and they all start in on the vein from the surface with the exception of the two lowermost, the 550 and the 650-feet levels; the first of which cross-cuts 40 feet through rhyolite before reaching the vein, and the second, 300 feet, chiefly through slate-shales.

It will be seen at once that the working of such a mine is a very simple matter. Each level is an adit to the surface, while the necessary winze-connections maintain excellent ventilation throughout the mine. The breaking of the ore is all done by overhand stoping, and the ore is handled entirely by gravity. All the ore above the 200-feet level is handled through it, winzes being connected past the 100-feet level by heavy timber shoots, and the ore being thrown directly from the zero to the 200-feet level.

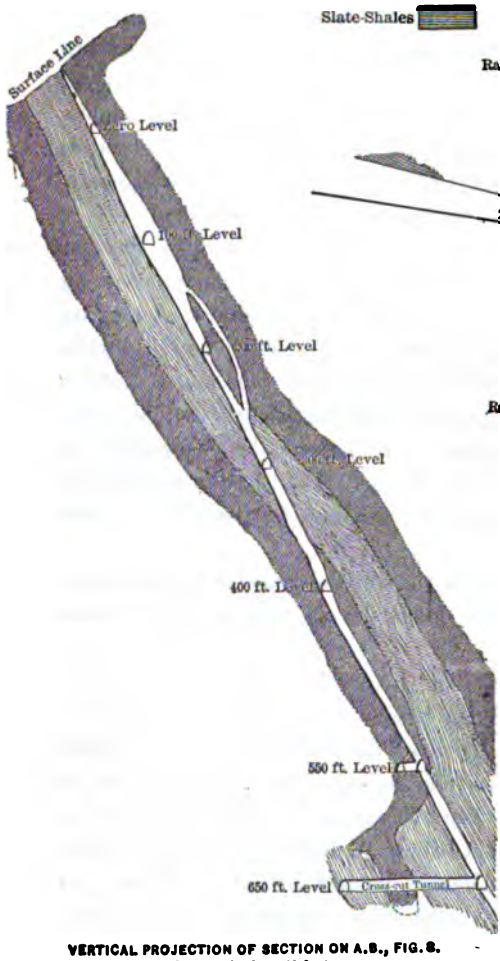
At the mouth of the latter is located a gravity-road or inclined plane, 800 feet long, with an average pitch of 40°, operated with iron skips of 1400 cwt. capacity. By it the ore is delivered to the Hallidie tramway, at the mouth of the 650-feet level. Ore extracted between the 200-feet and 400-feet levels is run out at the latter level and loaded into the skips by means of a bin with movable shoots, built over the gravity-road at that level. This road easily handles the 90 to 100 tons of ore extracted per 24 hours. Ore broken below the 400-feet level is run out at the tramway or 600-feet level.

A feature in the working of the mine worthy of note is the fact that all ore-extraction and much development-work, such as raising and sinking for connections, is done by native miners, American miners being chiefly employed in driving ahead the main levels. The native miner, properly handled, is no mean workman. Under the stimulus of a prize for the greatest number of feet drilled, some of them have sunk as many as four 2½ to 3 feet holes, in moderately hard stoping-ground, in a shift of 8 hours. Paid by the foot of hole drilled, they will earn $1.50 to $2.00 per day, but will do a corresponding amount of work.

With the introduction of air-drills for the more rapid advancement of exploration-work in the lower levels, more foreign miners will necessarily be employed.

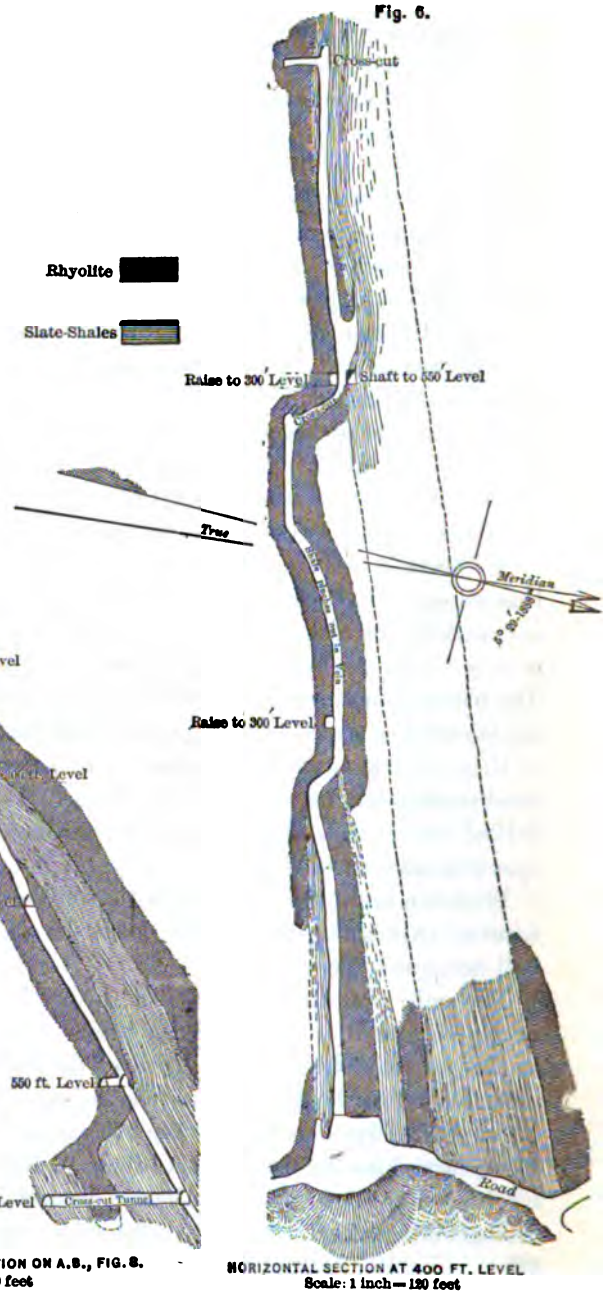
The only noteworthy feature of the compressed-air plant is the power employed to drive the compressor. It was found possible, by the use of a high head, to utilize the waters of the San Juan river, which in the dry season runs as low as 80 miner's inches (of 1.6 cu. ft.). Five hundred feet head was therefore adopted, the water being carried for 3000 feet in a 14-inch, 16 B. W. G. pipe to a bulkhead 500 feet above the wheel-house, from which it is led on to a 4-foot Pelton wheel, through a 12-inch wrought-iron pipe of Nos. 10, 9 and 8 B. W. G.; extra heavy iron being used on account of its greater durability. This water-power is sufficient to run a 60-ton concentrating mill as well, when needed.

Fig. 7.



VERTICAL PROJECTION OF SECTION ON A.B., FIG. 8.
Scale: 1 inch = 160 feet

Fig. 6.



HORIZONTAL SECTION AT 400 FT. LEVEL
Scale: 1 inch = 120 feet

With this drill-plant, development-work should readily be kept well ahead of extraction, and the lower levels, where the ground is too hard to drive by hand, opened up to give a large output of ore. All the indications are that, as these levels advance further into the mountain, the ore will become more base, and probably average less in gold and silver, though the latter point cannot yet be considered established, since in some cases high assays have been obtained from the undecomposed sulphurets. One small ore-body was encountered just above the 300-feet level, at about 400 feet in depth on the vein from the surface, and was smelted to a rich lead bullion. The ore consisted of galena, copper and iron pyrites, and assayed from 600 to 1000 ounces in silver and several ounces in gold per ton. This would indicate a likelihood of encountering bodies of high-grade base-metal sulphides in these levels.

Ore-deposition.

The question of the source of this ore-deposit becomes an interesting one in the light of the belief expresssd in reference to ore-deposits in general, by that eminent authority on the subject, Mr. S. F. Emmons,* who says of the usual vague statement that the ore came "*from below*," that, though possibly true, it "is generally inadequate and not founded upon sufficient study of the geological conditions."

Without venturing to enter into a discussion of this question with reference to the Rosario lode, I would briefly review the conditions to be observed in connection with the vein and mine, and let others judge as to their bearing upon a theory of leaching from the wall-rocks, such as that expressed by Professor Joseph Le Conte, who says:† "All speculators on this subject, I think, now hold that the mineral contents of veins are wholly derived by leaching from the rocks forming the fissure-walls."

Referring to Figure 8, it may be seen that,

1st. Above and west from the 550-feet level there lies an immense mass of eruptive rock.

2d. The major part of the work done in the mine to date lies in

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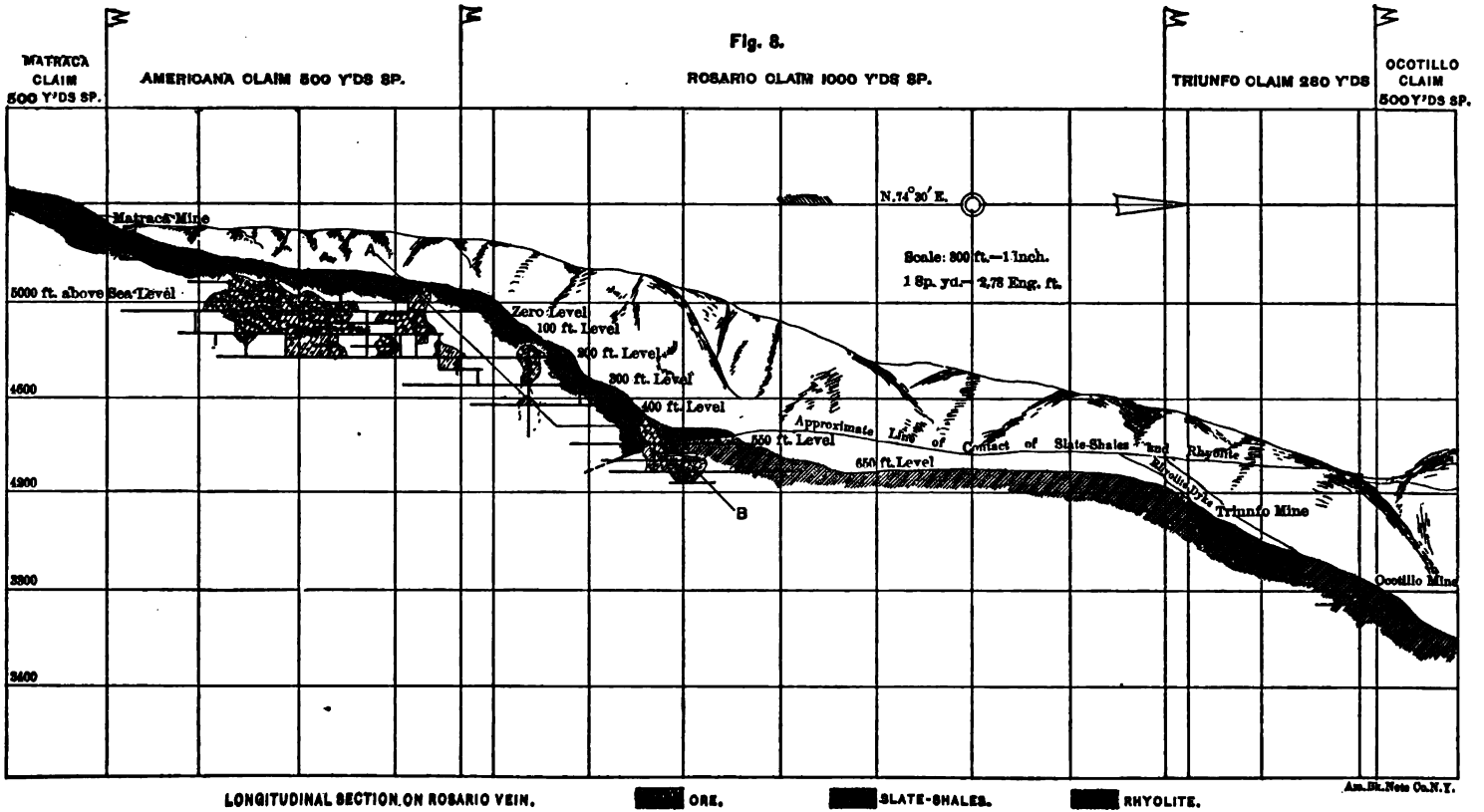
* *Engineering and Mining Journal*, May 12th, 1888, p. 339. Also, to the same effect, *U. S. Geological Survey*. Monograph XII., "Geology and Mining Industry of Leadville," p. 572.

† "Genesis of Metalliferous Veins," *American Journal of Science*, Third Series vol. xxvi., July, 1883, p. 6.

Fig. 8.


MATRACA CLAIM 500 Y'DS SP.
AMERICANA CLAIM 500 Y'DS SP.
ROSARIO CLAIM 1000 Y'DS SP.
TRIUNFO CLAIM 250 Y'DS
OCOTILLO CLAIM 500 Y'DS SP.
N.74°30' E.
Matraca Mine
Scale: 800 ft.—1 Inch.
1 Sp. yd.— 2.78 Eng. ft.
5000 ft. above Sea Level
Zero Level
100 ft. Level
200 ft. Level
300 ft. Level
400 ft. Level
550 ft. Level
650 ft. Level
Approximate Line of Contact of Slate-Shales and Rhyolite
B
Rhyolite Dyke
Triunfo Mine
Ocotillo Mine
4600
4200
3800
3400
LONGITUDINAL SECTION ON ROSARIO VEIN.
ORE.
SLATE-SHALES.
RHYOLITE.
Am. Bk. Note Co. N.Y.

this rock, and also the greater part of the ore extracted is from these upper levels.

3d. Considerable work has been done in the lower levels of the mine, all of which are at least 600 feet long, and the lowermost is entirely in slate-shales, where the vein has proven quite as strongly mineralized as in the upper levels; and from here a large body of several thousand tons of ore has been extracted. This is in fact the original mine, the portion of the vein worked by the natives before its acquisition by the present owners. There are other ore-bodies lying wholly in shale wall-rock, notably the first encountered in the 200-feet level, about 150 feet west of the mouth; but this rock has no thickness here, being merely the strip or wedge already described.

4th. Further to the east there are workings on the vein, 1200 and 2000 feet distant respectively from the Rosario tunnel, that are entirely in slate-shales, with the exception of a narrow dyke of rhyolite, that cuts through the bedded rock, and crosses the vein in the vicinity of the Triunfo mine, in much the same manner as the shale runs up into the eruptive rock; and in these workings the vein shows strongly mineralized, carrying copper pyrites and galena, but poorer in silver contents. This, however, may be due to nearness to the surface, as the workings are yet less than sixty feet in depth on the vein.

5th. The vein all along the surface here, through the sedimentary rock, shows a strong quartz outcrop, in places four feet wide, and is traceable down the mountain in the same rock for a long distance on the Ocotillo claim. Down in the gulch of the united San Juan and Escobales rivers, about 1600 feet below the mine, and half-a-mile distant from the nearest eruptive rock on the surface, there is a quartz-vein outcrop, and a drift run on it, lying on the strike, and appearing to be the continuation, of the Rosario lode.

6th. On the ridge shown in Fig. 8, lying back of and above the vein in slate-shales, there is at present several hundred feet thickness of rhyolite, while the quartz conglomerate and limestone that originally overlay the shales have been scored off.

7th. In this district both "formations" are seamed with many veins, the eruptive rock, being perhaps, the most prolific, or at least more discoveries having been made therein.

While in the mine there has been more work done in the upper levels (due largely to the faster progress to be made there, and the adaptability of the ore to the present mill), much more exploration-work will be done in the sedimentary rock with the air-drills now being put in, and with reasonable expectation of encountering

ore-bodies, to judge from the work already done, and the strength and continuity of the lode.

In regard to that portion of the vein lying wholly in rhyolite, there are but two facts that would militate against a theory of leaching of the vein-materials from the fissure-walls, and these are:

First. The great number of veins in the immediate vicinity of the Rosario lode, that, if filled in this manner, would tend to rob the adjoining rock and force a derivation of the materials from* "neighboring eruptive rocks ... not ... in actual contact with the ore-bodies," which undoubtedly exists here in plenty with depth; laterally the range is seamed with veins.

Second. This rock is remarkably destitute of the basic silicates, as mica, hornblende and augite, said by Dr. Sandberger to be the main rock-constituents furnishing the metals occurring in veins.†

It is noticeable that the leaching of rhyolite in the Yellowstone Park, described by Mr. Arnold Hague before the Montana Meeting of the Institute, in July, 1887, has been productive of no other metallic mineral than scorodite. This leaching has been going on for thousands of years, and is believed by Mr. Hague to be done by surface-waters, returned to the open air through the various vents, geysers, etc., after contact with the heated rocks below, the source of whose heat he ascribes to volcanic energy.

But a large part of the mineralized Rosario vein lies in sedimentary rock, and since "the eruptive rocks, rather than the sedimentary, are the more likely to carry the materials from which the vein-materials have been derived,"‡ it is necessary to assume that this vein-filling came from below.

Professor Le Conte says of the ascension theory,§ "According to this theory, as I understand it, the vein-matters, including the metallic ores, ... are gathered by leaching from the whole wall-rock, from top to bottom of the fissure; but mainly from the deeper parts, because these are under heavy pressure and superheat." In the case of the Rosario vein such leaching must have taken place at great depths, since above the vein were other sedimentary rocks that have been eroded, as already stated; while below these slate-

* "The Genesis of Certain Ore-Deposits," by S. F. Emmons, *Trans.*, xv., 138.

† *U. S. Geological Survey.* Monograph XII. "Geology and Mining Industry of Leadville," p. 576.

‡ "The Genesis of Certain Ore-Deposits," *Trans.*, xv., 145. Already cited.

§ "Genesis of Metalliferous Veins," *American Journal of Science*, Third Series, vol. xxvi., July, 1883, p. 7.

shales belong the lower red conglomerate and the Cantarranas-Peten limestone.

In order to fulfil, then, for this case, the conditions essential to a theory of leaching from the wall-rocks, it is necessary to assume in both formations great depth of fissure—a not unlikely factor.

But this approaches that suggestive "channel to the deep" alluded to by Dr. Raymond in discussing the Leadville ore-deposits,* while the lines here seem faint indeed that divide such theories from those of hot mineralized waters and general solfataric action, as advanced by Mr. J. E. Clayton to explain the method of vein-formation in the Drumlummon district.†

In short, it would seem quite possible that these vein-materials did "come from below," and the question of their source is reduced to speculation as to the manner in which they were taken into solution: whether by leaching at great depths, under heat and pressure, of a rock which does not contain the ingredients scientific research has found to be the chief source of the vein-metals; or (as in this case would seem more probable) by solution at great depths, under the volcanic conditions existent at the time of the eruption of the igneous rock.

SOAPING GEYSERS.

BY R. W. RAYMOND, NEW YORK CITY.

(Buffalo Meeting, October, 1888.)

SOME months ago I heard from a party of returned tourists an amusing story of a Chinese laundryman in the National Park who had included in his cabin a hot spring, of which he was accustomed to avail himself in his business, and who, on one occasion, having thrown a lot of linen into this spring to soak overnight, had added a piece of soap to facilitate the process, and had been considerably surprised when, stimulated by this soap, the spring had suddenly become a geyser and wrecked his establishment. The story went on to say that the practice of "soaping" geysers to make them active had become general, but had been forbidden by the authorities.

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* *Engineering and Mining Journal*, May 12th, 1888, p. 339.

† *Ibid.*, August 4th, 1888, p. 85.

To ascertain what basis of fact underlay this report, I wrote to our fellow-member, Mr. Arnold Hague, the geologist in charge of the National Park, and have received from him the following reply, which he authorizes me to incorporate in this paper. I regret that Mr. Hague's pressing duties in the field prevented him from preparing a paper on the subject; and I am glad that he will, at some future time, make public the results of his investigations, now in progress, concerning many points now somewhat obscure in the action of geysers, and particularly its relation to the element of temperature. The theory stated by Prof. Tyndall, in connection with the geysers of Iceland is, no doubt, a true one; but many local peculiarities of form, water-supply, mineral contents, and variations of temperature, may complicate, in a given region, the operation of the simple causes which that theory invokes. Much light may be expected from the careful and patient study of the subject which Mr. Hague is now making. Meanwhile, the preliminary account which his letter enables me to give, will doubtless prove interesting, especially to those of our members who enjoyed last year, in connection with the Utah and Montana meeting, the excursion through the Yellowstone National Park.

Mr. Hague writes as follows:

MAMMOTH HOT SPRINGS, Y. N. P., September 6th, 1888.

DEAR SIR:—In reply to your inquiries I may say that what is familiarly called "soaping geysers" has caused eruptions of several of the larger well-known geysers. I first heard of the action of soap upon the geysers early in the summer of 1885, from the late Chief Justice Waite, who had just returned from the Upper Geyser Basin, and whom I met at the Mammoth Springs. He asked me to explain the chemical action of soap upon the waters of the geysers, and told me that he had seen the eruption of a small geyser at the Chinese laundry. A few weeks later, in company with Mr. F. J. Haynes, photographer of the National Park, I made a number of experiments, which proved so satisfactory that I continued my investigations throughout the season on many of the hot springs and geysers in the principal basins. Mr. Haynes conceived the idea that for the purpose of photography it would be very desirable to be able to take negatives of geysers in action when the sun, clouds and winds were most favorable, and with the least possible loss of time to the photographer. Geysers which at their best are very fitful in their action, and only play at intervals of two or three weeks, consume much valuable time of any one attempting to make a large series of pictures. Discarding soap, we substituted Lewis's Concentrated Lye, put up in ½-lb. cans, which, as you may readily suppose, produced much more rapid action.

In a few words, the results of my investigations for the season, were that in one or two of the larger geysers eruptions could be hastened by the application of alkali, and that several of the smaller ones could be forced to throw out columns of water. Where the cauldrons or pools are large, but little change takes place, unless the waters are near the boiling-point. The most favorable conditions to accelerate

action are those in which the reservoirs are small, the vents or conduits through which the water is forced, narrow, and the water near the boiling-point. Of the large geysers, the "Beehive" seemed to be the one in which action could most easily be produced, although even here it fails in most instances. Here the water, for the greater part of the time, stands near the boiling-point. In the "Giant" geyser, which, at the time of my experiments, had not played for several months, I was able to cause most violent agitation and the throwing out of water, but nothing that could be called a genuine eruption. People familiar with the behavior of the "Giant" before an eruption, but ignorant of what I had done, believed the geyser was ready to resume its former activity. In the greater number of instances no action whatever was observed.

The boiling-point at the altitude of the Upper Geyser basin is about 198° Fahr. Many of the cauldrons and hot springs, however, when perfectly quiet, reach the temperature of 200° Fahr. before boiling. In many of these pools most violent action can be produced by simply throwing in a handful of fine gravel. Many of them boil up violently when stirred with a stick, and in one or two instances I have produced agitation, the water being thrown two or three feet, by simply lashing the surface with a rope. The "Giantess" geyser, which, as you may remember, is a large cauldron, constantly overflowing for days before an eruption, is very near the required temperature. This pool can be made to boil violently by simply stirring it with a cane, but has never been forced into an eruption by the use of soap or lye. I do not consider that there is any appreciable chemical action or increase of heat produced by the application of soap, but that the action is for the most part due to physical causes, inducing a boiling and bumping in the narrow vents, which in turn causes a relief of pressure and brings about geyser-action. I was soon able to predict with some certainty when eruptions could be hastened in this way. Many of the springs and pools present very much the same phenomena as are seen in the chemical laboratory where water is slowly heated in a beaker-glass by a Bunsen burner over a wire gauze. As you well know, solutions heated in this way frequently attain a temperature above the boiling-point without ebullition; but the moment a glass rod or a thermometer is plunged into the beaker violent agitation is brought about. It is also well known that solutions of caustic alkalies, before reaching a temperature of 212°, bump and boil in a very irregular manner, frequently causing a spattering and boiling over of the solution. I consider that the action of alkali in these reservoirs and cauldrons is very much the same. For some time I have intended publishing a short article upon the means of accelerating geyser-action, but have deferred it until I should have studied up the matter a little more, or until I was prepared to publish some other observations with regard to the relationship between temperature and geyser-eruption.

The line between a geyser and a hot spring is by no means an easy one to draw. It is a somewhat curious fact that the "Chinaman" geyser, where action is most easily caused by artificial means, has always been regarded by the Geological Survey as simply a hot spring, and none of our party have ever seen the "Chinaman" in action without the application of soap, except in one instance, when it was actually made to play to a height of 20 feet, after stirring it violently with a pine bough for nearly 10 minutes. In our records it is simply known as a spring, whereas by the application of soap a good geyser may be produced, throwing a column of water 20 feet or more into the air.

Attempts to "soap geysers" have become quite popular with tourists, so much so that I am told the toilet-soap is stolen in large quantities for this purpose from the rooms in the hotels. Throwing anything into the geysers and hot springs is now

very properly prohibited by the rules governing the Park. Where thousands of people visit the geyser-basins annually, it would of course be impossible to allow them to indulge their fancies by throwing all sorts of objects into the geysers. Most of the stories, however, which are circulated in regard to the eruptions produced by the action of soap are fabrications wholly without foundation; they may be put down as concentrated "lie." Very truly yours,

ARNOLD HAGUE, Geologist.

Mr. Hague is undoubtedly right in regarding the reaction in question as physical rather than chemical. Yet, the chemical effect is, it seems to me, the cause of the physical one; and I venture to suggest the following explanation in support of this proposition:

As will be remembered, the accepted theory of geyser-action is that immediately before an eruption, the water at each point in the geyser-tube has attained a temperature at which it is just prevented from flying into steam by the weight of the column of water above it. If that weight be diminished, an eruption is the necessary consequence. Such a diminution is brought about by the production or entrance (in spite of the weight of the column) of steam in the lower part of the tube, lifting the column and causing some of the water to overflow at the mouth of the geyser. Each particle of heated water, hitherto barely prevented from forming steam, is thus lifted to a level where its vaporization is practicable, and the more or less explosive eruption ensues. It is evident that the overflow is essential, although, as the case of the "Giantess," mentioned by Mr. Hague, indicates, an overflow is not always enough to cause an eruption. It must occur at the right moment.

The gradual entrance of steam below, and its quiet escape upwards, or its collapse in ascending, does not directly bring about the eruption. It merely effects the preparatory heating of the column. When this has been done a violent impulse, such as would be given by the generation or injection of a larger quantity of steam, seems to be required. Those who have stood by a geyser immediately before its eruption, will recall the abortive attempts which precede the final success. The successive collapses of steam below are evidenced by dull reports as well as by the "jumps" of the water in the tube. The boiling grows more and more violent; and at last, the supply of heat being everywhere sufficient, the water rises *until it overflows* and the geyser "goes off."

Whatever will produce violent boiling, then, may be expected to precipitate an eruption if the spring is ready for it, and even to cause an eruption in a spring which otherwise, boiling quietly, would not attain the eruptive conditions.

The effect of agitating the water by lashing the surface or by throwing in stones, is simply to liberate steam which has been retarded by the adhesion of the water. This sudden liberation may cause temporary violent boiling.

But I think the action of soluble salts thrown into the water is somewhat different. It is well known that mineral salts thus dissolved in water, raise its boiling-point. Yet nothing really boils but the water. The steam from solutions of non-volatile salts is as pure as if it had been generated from pure water. The action of the dissolved mineral, therefore, is a resistance to parting from the water. The boiling-point may be raised from 212° to over 300° Fahr., for instance, by saturating with calcium chloride. Ten per cent. of this salt raises the boiling-point nearly two degrees.*

Now, the effect of pouring lye into a geyser would be to form at once, on the top of the water-column, a more or less concentrated solution. This effect would be more rapid than that produced by soap, because the soap requires an appreciable time to dissolve. In both cases, however, it seems likely that only the water at and near the top would be affected, suffering an immediate rise of its boiling-point, or, in other words, presenting an increased resistance to the liberation of steam. This new hindrance acts like a lid on the boiling column, produces a more violent ebullition, and thus precipitates the eruption if the conditions are such that violent ebullition is all that is wanted for such a result.

In addition to this (especially in the case of soap) we have a viscosity of the liquid, hindering the escape of steam once formed, and thus producing an accumulation of bubbles, which hastens the overflow. I believe it is well known in the domestic laundry that the wash-boiler is liable to boil over when soap is added.

Finally, it has been suggested that the addition of caustic alkali to the water of a geyser might precipitate some of its ingredients, and they alter the character and behavior of the solution. If I am not

* See the results of Legrand's investigations, as given in Müller's *Physik*, 7th ed., Braunschweig, 1868, vol. ii., p. 738. The table does not include the caustic alkalies. I take calcium chloride as an illustration because of its strong hygroscopic properties. Of the salts included in the table above named, sodium chloride is the most effective in proportion to the percentage present, 7.7 per cent. being equivalent to 10 per cent. of the calcium chloride; but a saturated solution of the former at boiling-point contains but 39 7 per cent. of the salt, while the former may be thus dissolved up to 325 per cent. Considering the affinity for water possessed by the caustic alkalies, I think it safe to assume, for the present purpose, that 10 per cent. of potash would raise the boiling-point at least two degrees, and that in a concentrated solution of it the elevation would be at least 15 degrees, probably much more.

mistaken, however, the geyser waters are very dilute solutions, and do not carry any considerable proportion of bases which would be precipitated by an alkali. Moreover, any base thus precipitated would be replaced in the solution by its chemical equivalent of alkali. I think, therefore, that the final resultant effect on the boiling due to this cause may be neglected in the explanation.

COAL TRANSFER OF THE MT. CARBON COMPANY, LIMITED.

BY W. N. PAGE, POWELLTON, W. VA.

(New York Meeting, February, 1889.)

AMONG engineers engaged in mining coal for river transportation, probably no other subject of equal importance has received so little attention as the methods of transferring into barges and other craft.

The liability of all our streams to sudden freshets and their attendant drift and ice-gorges, seems to have had little effect upon the design and construction of what are commonly known as river-tipples, beyond the building of breakwaters, and other similar devices, to guard expensive structures from damage.

The first method adopted for such transfer was a simple incline, with a movable cradle, which could be raised or lowered to suit the different stages of water. This system, though limited to a small capacity, and to cars not larger than those commonly used underground, possessed two advantages. The first cost was comparatively small; and, since the structure did not obstruct the channel, there was little risk from freshets, the utmost damage from which could be repaired at small expense. Its disadvantages, however, were many, the principal among them being the breakage of the coal by reason of the necessarily high dump; the damage to barges from the same cause; and the difficulties of screening and distributing the different sizes into separate barges.

That system has been almost wholly abandoned of late, in favor of the basket-arrangement, lowering vertically into the barge, with the aid of counter-weights, which return the empty basket; while pipes or chutes convey the small coals into other barges, from a complete set of screens, which this method will readily accommodate. As far as my observations have gone, the latter system is now universally employed along all the tributaries of the Ohio. It

has been elaborated to meet enlarged outputs, until the cost of construction, in many cases, has reached $40,000 or $50,000. Where mines have been connected with the river by railway, necessitating the transfer into larger cars at the drift-mouth, the same system has been employed to lower the railway-cars, so as to avoid the breakage from an extra dumpage. Owing to the fact, however, that a barge must be loaded uniformly, to avoid damage, it has not been found advisable to dump more than two or three tons on any one point; and the largest car I have seen lowered in this manner had a maximum capacity of about six tons, dumping from the center upon a roofed cage, so as to distribute the load.

The Winifrede Coal Company has an arrangement of this kind, with the largest capacity on the Kanawha, excepting the transfer hereafter described. I believe its maximum work has been about 1300 tons in twelve or fourteen hours. Without knowing the exact cost, I would estimate it at between $25,000 and $30,000, including the structures for its protection. At the Faulkner mines, thirty miles above Charleston, a regular basket-arrangement has been erected, at a cost of about $26,000; and the Crescent mines, two miles lower, have a similar arrangement, costing about $20,000. I do not think either of these tipples can transfer more than 400 tons per diem. They were built by Pittsburgh contractors, and copied from the best devices in use there.

Since the best devised structures, upon such waters as the Ohio, are liable to be washed away, and become a total loss, it is clear that one of the first considerations should be economy in construction, and that as little resistance as the conditions will admit, should be opposed to the current. Where breakwaters are used, they must be raised, to afford absolute protection, at least fifty feet above low water. Such a structure in stone, would be beyond the means of most coal operators; and in timber, they are not only expensive, but depreciate rapidly at and above the water-line. Bearing the above conditions in mind, and confining my remarks to the transfer of bituminous coal into barges, I should name the following *desiderata* in order of importance.

1. The arrangement should permit a structure accommodating any car; and, since the majority of the mines, in the future, must reach the rivers over lines of railway too extensive for the economical use of small cars, the requirements of the standard railways should be met.

2. All bituminous coals, being more or less damaged by breakage, should be handled as gently as possible. Since breakage occurs not

only from the shock and friction on the bottom and sides of a chute, but from the weight and grinding of the superincumbent mass when in motion, it is clear that any method by which the coal can be literally carried, while it remains at rest, will best meet this condition.

3. The barge should be loaded uniformly; otherwise its timbers will spring, and leakage will become a source of constant expense. The cost of a barge is too great to permit the neglect of any precaution for its preservation.

4. The cost of transfer should be reduced to a minimum, and the machinery should not be so complicated for a large capacity, as to make a small output disproportionately expensive.

5. The capacity should be practically unlimited, since gravity furnishes unlimited power.

6. Above all, the structure should be a protection in itself against high water, or subject to such damage only, as may be quickly repaired at a small cost.

After a careful study of the conditions and requirements—which are of course slightly modified by circumstances—I have ventured to make a departure from the conservative methods now in use, and submit herewith drawings of a transfer erected on the Kanawha, for the Mt. Carbon Company, Limited, which I think are sufficiently clear to need little explanation.

As this transfer has been in use for several months past, I can refer to actual as well as theoretical results. As will be seen, it is a simple application of a flexible steel belt, 4 ft. wide, with 6-in. flanges, and 95 ft. from center to center of sprocket-shafts. The three chains are made of $\frac{1}{2}$ x 2-in. steel bars, with $\frac{3}{4}$-in. steel pins, 6 in. from centers. Each alternate link has a 2-in. flange, to which is bolted a segment of the belt, as shown on the detail-drawing; consequently every other link is an idler, which increases the flexibility of the belt. The segments are made of No. 12 gauge soft steel, and lap $\frac{3}{4}$ in. The lower lap is curved downwards, for which the idle link is notched.

The hopper underneath the car holds less than a ton; consequently the belt practically draws the coal direct from the car after this hopper has been once filled. The feed-apron is narrower than the belt, and is adjustable so as to feed automatically, the coal on the belt regulating the charge. I find no difficulty in regulating the feed to any desirable load; and no attendant is required. The links of the loaded chain run in small flanged rollers, placed every 4 ft. on 6-in. girders, 24 ft. in length, with fish-joints at every point of

suspension. It will be seen that there are four such points within the 95 ft., for which I have used eight 2-ton differential blocks, four on each side. I first proposed to use counterweights at these points; but upon estimation found the blocks were the cheaper and more satisfactory.

The total weight of belt and attachments, exclusive of the upper, or fixed, shaft and sprockets, is five tons; and the level load is five tons; consequently each revolution of the belt conveys a little more than ten tons. The arrangement for returning the empty portion of belt consists of three pulleys, 2 ft. in diameter, by 12 in face, set on the shaft so that each pulley supports a chain. To suit the different stages of water, the entire system can be readily raised or lowered at one end, by one man, care being taken to have the girders in line when the proper elevation is reached. The guide on the movable end is an arc of the circle to which the belt is radius; though, to cover any inequalities, or spring in the timbers, the fixed ends of the girders are provided with movable connections to the main shaft, which is also provided with extension-boxes, to facilitate making the connections. The chute which distributes the load in the barge is built of light steel, arranged so as to convey the coal to both sides at the same time, the proportion delivered to either being regulated by the angle of inclination. It is adjusted so that the coal moves in it very slowly, and has a minimum drop into the barge. When the sides have been fully loaded, the hinged apron is folded back, and the discharge falls in the center of the barge, which is thus loaded so as to require little or no levelling.

It is apparent that the greatest inclination of the belt should not exceed the friction-angle of coal on iron; and that, within certain angles, gravity will revolve the belt, on the principle of an ordinary incline. Though I put a brake-wheel on the main shaft, intending at the time to use gravity-power, I determined later to make use of a small 6 x 8-in. engine, which was on hand to operate a capstan for handling the barges. I connected the belt with it, and made the greatest angle of inclination only 15 degrees. The belt is speeded to 100 ft. per minute—the engine making 150 revolutions—and makes a complete revolution in a little less than two minutes. In actual practice, the belt empties a twelve-ton car in two minutes, or at the rate of 360 tons per hour. This capacity can be increased, without additional expense, by simply increasing the speed of the belt, or, at a very small additional cost, by increasing its width.

As the belt has never been operated by gravity alone, I am not able to say exactly what the factor of friction is, but am satisfied

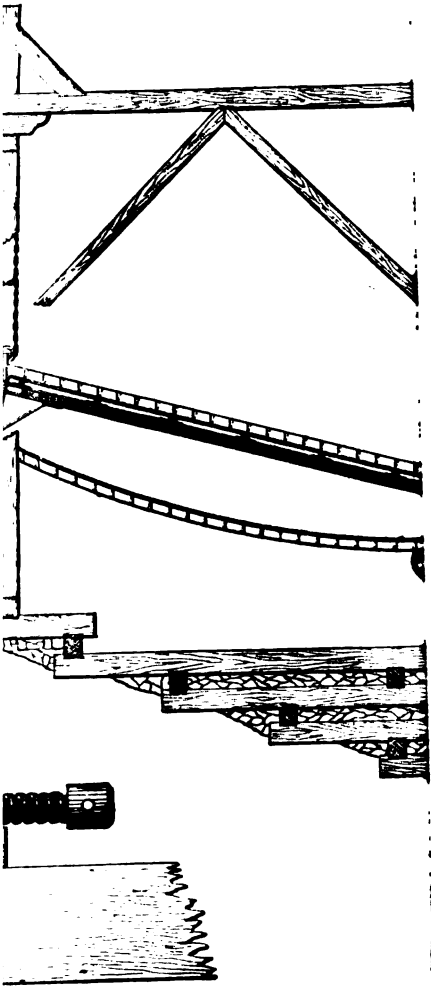
that it will revolve between the angles of ten and thirty degrees; in other words, it can be operated by gravity alone, through an arc of twenty degrees; and with a radius of 95 ft. this arc would approximate a difference in elevation of 34 ft., which is more than ample to cover all loading-stages in the river.

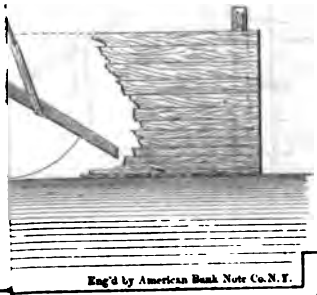
I find that the little engine mentioned will run the loaded belt without difficulty, in a horizontal position; consequently the total friction must be considerably less than the power of its cylinder, as it is geared (the gravity force exerted in direction of belt, at any angle, may be easily determined by the resolution of forces). The total cost of all the iron in the structure was $2400, excepting only the engine and boiler, which cost originally about $600. To this must be added 100,000 ft. of white-oak timber in the crib and trestle, which cost $2000 erected, and $600 for stone filling, making the total cost, in working order, $5600, including engine, capstan, and all gearing. The cost in timber and stone will, of course, vary according to conditions. It will be seen from the drawings that I have used timbers lavishly, both in the wings of the crib and for the protection of the bank. The whole structure is so framed and drift-bolted together that no part can be moved independently of the other, including the stone crib, which extends back to the railway grade, fifty feet above low water, and ten feet above the Chesapeake and Ohio Railway.

The distance between bents being 24 ft., little resistance is offered to the current (none of them are in the channel); and even should drift or ice gain a lodging, there is as much strength to withstand the pressure as is presented by an ordinary breakwater. The belt can be raised clear of water, up to a 40-ft. rise, and should this (the highest gauge yet recorded) be exceeded, it can be lowered in safety beneath the surface. As the crib is filled so that it cannot be undermined, the greatest damage which could possibly result would be the breaking of the bents, which are composed of four timbers of 12 x 12-in. oak, each firmly bolted and braced to the stone crib and its two wing-walls. The belt would of course remain; and even such an improbable catastrophe would result in small damage.

In actual working, only two men are required to run this conveyor to its full capacity: an engineer, at $1.50 per day, who also attends the capstan, and one man on the barge at $2 per day, who is a carpenter and calker; consequently the total cost per diem does not exceed $3.50. The trainmen dump the cars; but if this expense be added it would amount to $2.50 more. The tracks are arranged so that the loads run in, and empties out, on down grades

Fig. I.





Eng'd by American Bank Note Co.N.Y.

In three months' working the cost of repairs has been nothing; and since every member has a large bearing and safety-factor, the wear and depreciation will be very slight. No breakwaters have been put in, nor is there need for any; since the current is slacked by Dam No. 2.

Apart from the advantages already named, it has been clearly demonstrated that both coal and coke can be put in barges with half the breakage of any other arrangement I have seen; that there is nothing complicated to get out of order; that, as the barge sinks in proportion to the load, the belt has to be moved only for the different stages of water. Moreover, there is no danger to life or property from the breakage of any part, as is the case with the vertical system, in which I have frequently known the breaking of a rope to cause the loss of life, barge and car.

With self-dumping cars, I would feel safe in guaranteeing to transfer 4000 tons with this machine, within twelve hours, at a cost per ton not exceeding two-tenths of a cents, and, with a few hundred dollars extra outlay, I would undertake to double this capacity, and lower the cost named.

As the mines of the Mt. Carbon Company are connected with the river by five miles of standard-gauge railway, and their coal is screened at the mine-tipple, where they have 202 bee-hive ovens, no arrangement has been made to separate the nut and slack at the river. The vertical distance between the car and belt, however, could be easily utilized for screens; and the small coals could be conveyed to either side of the belt, where they could be carried by a trough and scrapers, connected to the main shafts, into separate barges, which might be arranged inside of the lump-receiver. Under favorable conditions, the angle of inclination to these inside barges would be sufficient to slide the coal in fixed pipes, since breakage there need not be considered.

In submitting this transfer to the Institute, I am convinced that the principle is the correct one, and that it is capable of modifications to meet the greatest number of conditions, for various purposes other than the handling of coal. To those who may have any doubts, I will take pleasure in pointing out actual results. An engineer of reputation, manager of a large property, and prepared to doubt all things new, recently stood for several hours with watch in hand, timing the cars as they were dumped. When satisfied, he informed me that the rate of transfer, by his time, was 500 tons per hour. Yet, in addition to the brakemen who were dumping the cars, there were only two workmen present, the engine-man and the barge-man.

PIG-IRON OF UNUSUAL STRENGTH.

BY FRED. P. DEWEY, WASHINGTON, D. C.

(Buffalo Meeting, October, 1888.)

THE product of the Muirkirk, Md., furnace has always enjoyed a very high reputation for strength; and this is supported not only by its behavior in practice, both alone and in mixtures, but also by the following series of tensile tests, made at various times during the last fourteen years:

South Boston Iron Co., November 10th, 1874.

Grade.	Strength. Pounds per square inch.	Density.
No. 3, ...	30,685	7.2794
No. 4, ...	35,140	7.2713

Washington Navy Yard, March 31st, 1875.*

Number of sample.	Shape of sample.	Diameter of sample. Inches.	Length of sample. Inches.	Breaking-strain in pounds.	Breaking-strain in lbs. per sq. inch.	
1	Groove.	.568	2.25	5275	20817	No. 2 Iron. Average, 21601 lbs.
2	"	.567	2.25	5600	22178	
3	Curved groove.	.568	groove.	5350	21113	
4	" "	.568	"	5650	22297	
1	Cylinder.	.568	2.25	5690	22454	No. 3 Iron. Average, 23959 lbs.
2	"	.568	2.25	6460	25493	
3	Curved groove.	.568	groove.	5655	23316	
4	" "	.568	"	6480	25572	
1	Cylinder.	.648	2.4	15750	47756	No. 4 Iron. Av., 41329 lbs.
2	"	.660	2.5	13750	40193	
3	Curved groove.	.660	groove.	14425	42166	
4	" "	.665	"	14000	40311	Sample. Sp. gr.
5	Cylinder.	.564	2.25	9920	39712	No. 1, 7.3251
6	Groove.	.564	groove.	9460	37840	" 2, 7.3467

Philadelphia Scale and Testing Works, October 31st, 1879.*

Samples marked.	Diameter in inches.	Area in sq. inches.	Breaking strain in lbs.	Breaking strain in lbs. per sq. in.	
S	0.798	0.5	20380	40760	No. 4 Iron. Average, 40771 lbs.
S	0.804	0.5076	21100	41568	
H	0.802	0.5051	20740	41061	
H	0.802	0.5051	20050	39695	

* *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. v., p. 47.

Atlas Works, Pittsburgh, Pa., May,* 1882.

	Breaking-strain in pounds per sq. inch.
Tested on Gill's testing-machine, No. 4 iron, . . .	42,300

Thomas Iron Co.,† Hokendauqua, Pa., November 16th, 1883.

	No. 4 iron. Breaking-strain in pounds per square inch.
1st, ...	50,320
2d, ...	52,160
Average, ...	51,240

All these tests were made upon pieces cut from the pig as cast in the usual way at the furnace.

That this strong iron is not accidental is shown by the fact of its being produced during so many years.

At first no break was made in the furnace-grading between No. 4 and mottled iron, but it was observed that the high tests were always obtained with a very close and uniform iron, so that now two and sometimes three grades are made between Nos. 4 and 5, and on special order the pig is so graded that 40,000 pound metal is sold with a guarantee. The last three tests given above were of iron selected in this way, and no iron so selected has ever fallen short.

The following are recent tests made at the Watertown Arsenal, and, while the figures are lower, it must be remembered that this machine gives uniformly lower results.

U. S. Testing Machine, Watertown Arsenal, Mass., March 5th, 1885.
Four Samples Muirkirk Pig-Iron by Tension.

Test numbers.	Furnace grade. No.	Diameter. Inches.	Sectional area. Sq. inches.	Ultimate strength.		Appearance of fractures.
				Total lbs.	Lbs. per square inch.	
3983	2	1.132	1.006	19,040	18,930	Coarse granular.
3984	3	1.137	1.015	22,950	22,610	Varies from fine to coarse granular.
3985	4	1.137	1.015	32,840	32,350	Fine granular uniform appearance.
3986	4½	1.140	1.021	35,010	34,290	Fine granular with light gray spots.

Total length of specimen, 23 to 23¾ inches. Length between collars, 16¼ to 17 inches. No. 3983, fractured at neck. No. 3984, fractured at middle of stem. No. 3985, fractured 1 inch from middle of stem. No. 3986, fractured at neck.

* *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. iii., p. 185.
† *Ibid.*, vol. v., p. 47.

Recently a complete series of specimens, consisting of two each of Nos. 4, 4¼, 4½ and 4¾, were sent to the Yale and Towne Co., with the understanding that they would test each number as received, and after annealing 96 and 144 hours, on an Emery machine, similar to the Watertown Arsenal machine. For some unexplained reason the full plan was not carried out by the Yale and Towne Co., and the results are not nearly as valuable as they would be had this been done; but they present some interesting and valuable points and I therefore give their report in full:

Report of Tests of Muirkirk Charcoal Pig-iron, by the Yale and Towne Manufacturing Co., Stamford, Conn., Oct. 8th, 1888.

Grade.	Before annealing.		Annealed 96 hours.		Annealed 144 hours.	
	Breaking load in lbs. per sq. in. in tension.	Character of the iron as shown by fracture.	Breaking load in lbs. per sq. in. in tension.	Character of the iron as shown by fracture.	Breaking load in lbs. per sq. in. in tension.	Character of the iron as shown by fracture.
No. 4.	32,287	Gray and close.	24,181	Dark, and more open than before.		
	33,074	Gray and close.	24,241	Dark, and more open than before.		
No. 4¼.			26,482	Dull dark-gray, with some silvery gray.	26,775	About the same as after 96 hrs. anpealing.
			25,197	Dull dark-gray, with some silvery gray.	27,564	About the same as after 96 hrs. annealing.
No. 4½.			33,919	White iron with spongy center.	37,798	White with spongy center.
			40,855	White with no flaws.	A very bad flaw.	A very bad flaw.
No. 4¾.			41,025	White with spongy center.	38,217	White with no flaw.
			36,893	White with spongy center.	34,471	White with spongy center.

NOTE.—In each case the pig-iron was melted in a graphite crucible and cast in green-sand mould, in test-bars 18 inches long by 1 inch in diameter.

Compared with the above figures the following table gives the tensile strength of similar irons made with charcoal for fuel, and with a warm blast:

	Pounds per square inch.
Stickney Iron Co., No. 4,*	30,000 to 33,000
Salisbury Iron Co., No. 4,†	34,407
Tecumseh Iron Co., No. 4,‡	27,800
	Density.
Copake Iron Co., No. 4,§	7.3347 — 33,428
" " " " 4½,	7.3131 — 28,627

The Muirkirk Furnace|| is located at Muirkirk, Prince George's county, Md., 14 miles from Washington, D. C., and 26 miles from Baltimore on the Baltimore & Ohio R.R. The stack is 29 by 8½ feet in size, and was built in 1847. It is worked with open top, and the blast is heated by a Raymond & Campbell stove. The blowing-engine has two cylinders, 5-feet stroke, 42 inches diameter, running about 9 revolutions per minute, and the blast is delivered at ⅝ to ¾ inch of mercury.

Its capacity is 4200 tons yearly, and the product is mainly consumed for car-wheels and chilled work generally, for armament, for flanged iron, and for high-grade castings, for which its strength eminently fits it. It is owned and operated by Mr. Charles E. Coffin.

The ores used are generally known as Baltimore ores, and consist of carbonate of iron, sometimes found pure in large masses, but generally more or less altered to limonite and hematite. Sometimes the alteration is complete, and no trace of the carbonate remains. It is not infrequent to find pieces of natural charcoal in the carbonate-ore, but a search extending over several years has failed to discover any in the solid altered ores. Occasionally beautiful small crystals of selenite are found.

The ores are classified roughly as white and brown ore, the first consisting largely of the unaltered carbonate, which is very light-colored, while the second consists largely of the altered carbonate, being partly limonite and partly hematite. Some of the furnaces of the region esteem the white ore above the brown, but there does not seem to be

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* See beyond.

† *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. v., p. 366.

‡ *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. v., p. 36.

§ MS. in National Museum.

|| *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. i., No. 2, p. 10.

any real ground for so doing. The ore occurs in scattered nodules, in clay of the upper division of the Potomac formation of McGee,* the exact age of which is in doubt, but it is either very late Jurassic or very early Cretaceous, the fauna† indicating the former and the flora‡ the latter. There is more or less apparent banding in the position of the nodules, which corresponds with the bedding of the clay.

These ore-bearing clays extend from just north of the Susquehanna river to the Potomac river, but their greatest exploitation is about the city of Baltimore, whence the name Baltimore ores.

The general plan of mining is the same in the whole region. Wherever ore is found, or the indications seem to be good, digging is commenced, and the ore found is taken out. This is continued in an open cut as long as sufficient ore is found to pay for removing the cover. When the cover becomes too great the opening is abandoned, and a new one is made elsewhere. The old miners say that an inch in thickness of ore will pay for removing a foot thick of clay, and it is found in practice that from 10 to 12 loads of clay are taken out for one load of ore. A bank producing 10 tons daily is considered a good one. Regular mining has been undertaken in a few cases, but has been soon abandoned, either completely or for open-cut work, based upon the underground work. A peculiarity of these clays is that the dumps from the ore-banks form a congenial soil for white clover, which springs up spontaneously, and in a single season will form a very fair cover, thus changing the unsightly dumps into a pleasing feature of the landscape.

The composition of the ores is shown by the following analyses:

Average of Ten Analyses made by J. Blodgett Britton in 1868 *and* 1869:

	Per cent.
Metallic iron,	45.38
Insoluble siliceous matter,	15.30
Sulphur,	0.078
Phosphorus,	0.119

The loss by ignition averaged for three determinations 12.80 per cent.

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* *Am. Jr. Sci.*, Third Series, vol. xxxv., p. 129:

† O. C. Marsh, *Am. Jr. Sci.*, Third Series, vol. xxxv., p. 89.

‡ L. F. Ward, *Am. Jr. Sci.*, Third Series, vol. xxxvi., p. 119.

*Analyses from Tenth Census Report.**

	Carbonate, Hematite and Limonite. G. L. Skaggs, Branchville.	Carbonate and Hematite. Geo. Yokel, Beltsville.	Hematite with some Limonite. Muirkirk furnace, Prince Geo. County.	Carbonate. Furnace-sample, Muirkirk furnace.	Hematite with a little Carbonate. J. O'Brien, Contee's.
	No. 1037.	No. 1038.	No. 1041.	No. 1042.	No. 1043.
Metallic iron	40.85	46.51	43.25	41.07	45.57
Phosphorus	0.060	0.115	0.101	0.060	0.090
Phosphorus in 100 parts of iron	0.147	0.247	0.233	0.146	0.197

REMARKS.

No. 1037. Sample taken from a car-load of 20 tons, Branchville station, B. & O. Railroad.

No. 1038. Sample taken from pile of about 10 tons of nodules at the mine.

No. 1041. Sample taken from pile of about 5 tons at Muirkirk furnace, 2 tons on carts, 3 tons at the ore-banks.

No. 1042. Sample taken from pile of about 1000 tons, Muirkirk furnace, white or strictly carbonate portions selected; raw ore supplied to furnace by Freil & Millbrook's, Donaldson's and Wood's banks.

No. 1043 Samples taken from 30 tons of nodules at the mine and 40 tons at Contee's station.

	Carbonate. Hematite, Limonite. Furnace sample, Muirkirk furnace.	Roasted ore. Furnace sample, Muirkirk furnace.	Remarks
	No. 1039.	No. 1040.	
Metallic iron	42.91	46.25	No. 1039. Sample taken from pile of about 1000 tons at Muirkirk furnace.
Phosphorus	0.073	0.108	
Phosphorus in 100 parts iron	0.170	0.234	

* *Tenth Census*, vol. xv., Mining Industries, pp. 248 and 546.

Analyses by the Chemist of the Thomas Iron Company.

Muirkirk ores.	Protoxide of iron.	Peroxide of iron.	Alumina.	Lime.	Magnesia.	Oxide of Manganese.	Silica.	Phosphoric acid.	Phosphorus.	Metallic iron.	Water.	Water and Carbonic Acid.	
No. 1.		70.57	6.38	1.22	.52	5.02	10.44	.219	.095	49.40	5.64		Brown ore, Walker Bank.
" 2.		77.80	1.49	1.58	1.27	4.20	5.84	.633	.277	54.46	7.19		Brown ore, Walker Bank.
" 3.	49.10		3.35	2.12	.66	2.06	6.42	.185	.080	38.1		36.11	White ore, Millbrook.
" 4.	58.39		2.47	2.32	1.09	4.85	4.54	.215	.094	45.42		26.13	White and brown ore, Millbrook.
" 5.		74.01	3.92	1.76	3.66	1.20	4.80	.79	.348	51.81	9.86		Brown ore, Millbrook.
" 6.		68.15	7.16	.69	.31	3.05	15.28	.118	.051	47.71	5.25		Top ore from Diven.
" 7.	47.09		3.97	2.85	1.72	5.26	10.10	.143	.062	36.63		28.87	Branchville, Burgess.
" 8.	47.25		3.14	3.38	1.69	6.29	7.96	.188	.082	36.75		30.11	Branchville, Burgess.
" 9.	53.75		2.20	2.64	.85	5.30	4.42	.262	.114	41.81		30.53	Jefferson, Tyson.
" 10.	50.65		2.42	1.40	1.59	3.98	7.62	.106	.046	39.40		32.24	Montieth.
" 11.	51.55		5.11	1.92	.49	5.33	13.28	.239	.104	41.10		22.09	J. Collins's Bank.
" 12.	52.35		2.49	2.82	1.73	3.06	3.76	1.26	.55	40.72		32.53	Dennis Collins's Bank.
" 13.	52.20		3.19	2.90	1.19	3.49	3.98	.77	.33	40.60		32.28	Dennis Collins's Bank.

All the ore is roasted before smelting, so that practically all the iron enters the furnace as anhydrous sesquioxide of iron. Circular kilns are used for roasting; charcoal and coke-braize are used for fuel; but no account of the amount consumed for this purpose has ever been kept. The fuel used in the furnace is charcoal, made principally from pine, chestnut and oak. Formerly only pit- or meiler-coal was used; but in the latter part of 1882 three circular kilns were erected at the furnace. The kiln-coal proved so satisfactory that more kilns were erected; and now almost all the charcoal is made at the furnace in these kilns.

Oyster-shells, obtained from the oyster-dealers in Baltimore, are used for flux. They make an excellent flux, requiring no crushing or other preparation, and are high in carbonate of lime, as is shown by the following analysis of an air-dried sample made by J. Blodgett Britton:

	Per cent.
Carbonate of lime,	96.45
Carbonate of magnesia,	.85
Insoluble earthy matter,	1.34
Sulphuric acid,	.07
Phosphoric acid,	.03
Alumina with a little oxide of iron,	.45
Water, soda, undetected matter and loss,	.81
	100.00

The ordinary running of the furnace is illustrated by the following records:

Muirkirk Furnace, Md.*

Average weekly product of pig-iron for six weeks, in tons of 2240 pounds,		80.2 tons
Estimated carbon in fuel,		90.0 per cent.
Average heat of blast,		850 degrees
Average quantity used to the gross ton of pig-iron made.	*a.* Fuel,	2410 pounds
	b. Iron ore,	4827 "
	c. Limestone,	690 "
Burden (total ore and limestone), per ton of iron,		5517 "
Burden, per pound of charcoal,		2289 "
Burden, per pound of carbon,		2544 "

* *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. ii., No. 6, p. 373. From a comparison of the operations of twelve typical charcoal furnaces.

Record of Muirkirk Furnace, Md.

Record of Muirkirk Furnace, Md.	Blast No. 17.* April 7 to Dec. 26, 1882.	Blast No. 22. Aug. 15 to Oct. 31, 1887.	Blast No. 23. May 14 to July 31, 1888.	
In blast	264.5	77	78	Days.
Stopped	30. *a*	0	3.25*b*	Days.
Actual running	234.5	77	74.75	Days.
Product of charcoal pig	2,584.5	1,137	1,157	Tons.
Amount of ore used	12,192,705	5,346,780	5,546,083	Pounds.
Amount of charcoal received at furnace†	296,229 *c*	123,325 *d*	128,613 *d*	Bushels.
Amount of ore per ton of iron.	4,718	4,702	4,793	Pounds.
Average charcoal to ton of pig.	114.6	108.5	111.16*e*	Bushels
Best week's work (7 days consecutive)	96	110.	124.5	Tons.
Best three weeks' work (consecutive)	273	324.	355.5	Tons.
Best week's work on charcoal (7 days consecutive)	97.23*f*	101.68*g*	98.99*h*	Bushels.
Best three weeks' work on charcoal, consecutive (273 tons)	104.85	.....	.....	Bushels.
Average day's work for blast	11.02	14.76	15.34	Tons.
Oyster shells, as flux, per ton of iron	950	877	838	Pounds.
Revolutions of blowing cylinder, from 5½ to 9 per minute.	.....	.....	.....	...

a. For charcoal, not continuous.

b. For repairs.

c. From one-third oak and chestnut, and two-thirds pine. Pit- or meiler-coal (2748 cubic inches per bushel).

d. Mostly kiln-coal (20 pounds to bushel).

e. July 31, 1888, bridge-house, boiler-house, etc., destroyed by fire. All charcoal in stock is charged to blast, which accounts for the high average.

f. On 96 tons of iron.

g. On 104 tons of iron.

h. On 124.5 tons of iron.

It was hoped that the last blast, No. 23, would bring the fuel-record down to 100 bushels of charcoal per ton of iron, and a good start was made in that direction; but the unfortunate conflagration which prematurely cut short the blast, raised the average above the previous blast.

The 1157 tons of iron made in this blast were graded as follows:

Grade.	Tons.	Per cent.
No. 2,	45.5	3 93
No. 3,	84	7.26
No. 4,	346	29.91
No. 4¼,	223.5	19.32
No. 4½,	80	6.91
No. 4¾,	137.5	11.88
No. 5,	81.5	7.04
No. 6,	159	13.74

* *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. iv., No. 1, p. 22.

† Includes all waste.

These figures represent fairly the average normal product of the furnace, and show well the tendency to make close (high) gray irons. While only 11 per cent. of the iron was graded below No. 4, less than 21 per cent. was graded Nos. 5 and 6, so that considerably over two-thirds of the iron was No. 4 and its fractions.

No. 2 iron is a rarity at the furnace, and while No. 1 is not unknown, yet it is seldom produced. The chief reason for this is that the production of No. 1 iron unduly increases the fuel-consumption.

The 583 tons of iron made in the Wassaic* furnace, from all carbonate-ore, were graded as follows:

Grade.	Tons.	Per cent.
No. 3,	15	2.57
No. 4,	442	75.81
No. 4½,	99	16.98
No. 5,	16	2.74
No 6,	11	1.89

These figures show, even more markedly, the same tendency to high gray irons; and Mr. Gridley states that he found it impossible to make soft irons. Although he succeeded in making what would normally be a No. 1 cinder, yet the iron was always close.

In making the iron from which the No. 2 test of the Thomas Iron Company was taken, 114 bushels, equal to 2280 pounds, of charcoal per ton of iron were used, and the flux was 20 per cent. of the weight of the ore. The raw ore averaged 36.4 per cent. in iron. Twenty charges of 1200 to 1600 pounds of ore were made per twelve hours.

The following analyses have been made of this metal:

No. 1. *Muirkirk Pig-Iron No.* 4.

Carbon,	2.93
Silicon,	0.72
Sulphur,	0 035
Manganese,	0.93

No. 2. *Muirkirk Pig-Iron No.* 4.

Phosphorus,	0.274
Silicon,	0.83
Manganese,	0.95
Sulphur,	0.047

REMARKS.

No. 1. From drillings made in cutting out the test-pieces. Analyst, the chemist of the Thomas Iron Company.

No. 2. Drillings taken from the test-piece after fracture and analyzed by Mr. J. A. Allen under the writer's direction.

* *Trans.*, xii., 521. See also beyond.

The slag made at the same time showed the following composition:

Muirkirk Slag.

	Per cent.
SiO_2,	46.85
Al_2O_3,	12.84
FeO,	3.20
MnO,	7.53
CaO,	25.35
CaS,	.34
MgO,	2.11
K_2O,	2.23
Na_2O,	.49
	100.94

Analyst: J. A. Allen.

In constructing the large pumping-engine for the Pittsburgh, Pa., water-works, the Atlas Works* had great difficulty in getting a suitable iron for the valve-chambers, etc. After investigating the Muirkirk iron they determined to make use of it, after which no more trouble was experienced.

During the investigation the test of strength quoted on p. 461 was made, and also the following analyses of strong irons, which show wide variations in graphite, combined carbon and manganese.

Muirkirk Pig-Iron No. 4.

	September 12th, 1882. Per cent.	June 19th, 1883. Per cent.
Silicon,	.782	.919
Phosphorus,	.244	.276
Graphitic carbon,	2.55	3.080
Combined carbon,	.155	.650
Manganese,	.565	1.441
Sulphur,	.271†	.016
Iron,	95.433	93.618
	100.000	100.000

Analyst: William Graff, of Graff, Bennett & Co.

Other analyses of Muirkirk iron are:

No. 4½.

	Per cent.
Silicon,	.66
Manganese,	1.38

Analyst: J. A. Allen.

* *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. iii., p. 185.

† The decimal point here is evidently misplaced, and it should be S, 0.027. Of many determinations of sulphur in this iron that I have, no other is over 0.05 per cent.—F. P. D.

No. 4 Re-melted.

	Per cent.
Silicon,	.633
Phosphorus,	.247
Manganese,	.820

No. 4½ Re-melted.

	Per cent.
Silicon,	.613
Phosphorus,	.221
Manganese,	.666

Analyst: A. W. Whitney, Chemist of the Whitney Iron Works.

High Chilling-Iron, Giving ¾-Inch Chill in Pig-Bed.

	Per cent.
Phosphorus,	0.22
Silicon,	0 59
Manganese,	1.52

Analyst: Chemist of Pennsylvania Railroad Company.

Other furnaces in the region working upon the same ores, with the same fuel and flux, and with all the conditions of running practically the same, have not succeeded in making so strong an iron.

By the kindness of Mr. G. H. Stickney, president of the Stickney Iron Company, I am enabled to give the following results from their records of 1882. These furnaces are situated on the outskirts of Baltimore:

Roasted Baltimore Ore Used by the Stickney Iron Company, Making Warm-Blast Charcoal-Pig for Car-Wheels, etc.

	Per cent.
Sesquioxide of iron,	65.394
Alumina,	7.204
Oxide of manganese,	2.740
Silica,	20.460
Phosphoric acid,	0.397
Sulphur,	trace
Iron,	45.77
Manganese,	1.97
Phosphorus,	0.173

Analyst: George H. Stickney.

Analysis of Oyster Shells, Stickney Furnace.

	Per cent
Calcium carbonate,	94.48
Magnesium carbonate,	0.94
Ferric oxide, Alumina,	0.53
Sulphuric acid,	0.80
Phosphoric acid,	0.06
Soda,	0.65
Insoluble,	2.54

No. 4 Pig of the Stickney Iron Company.

	All Maryland	Part Maryland. Part Foreign.	All Foreign.	
	Furnace A.	Furnace B.		Remarks.
Silicon	0.774	0.572	0.829	Booth, Garrett and Blair, Analysts.
Sulphur	0.044	0.063	0.051	
Phosphorus	0.282	0.229	0.069	
Manganese	0.834	0.715	0.368	
Carbon	.....	.....	3.256	

Tensile strength No. 4 iron, 30,000 to 33,000 pounds per square inch.

At the Roanoke Meeting of the Institute, Mr. E. Gridley* described some pig-iron† made in the Wassaic furnace, Amenia, Dutchess county, N. Y., which was unusually strong, the average of thirteen tests of No. 4 iron giving 41,349 pounds.

Mr. Gridley‡ afterwards reported two more tests, which gave 44,500 and 47,500 pounds tensile strength per square inch. Still later§ he reported four more tests, running from 42,254 to 48,400 pounds tensile strength per square inch, and also gave two analyses of the raw carbonate-ore.

This iron was also made from a carbonate-ore, but richer in iron and manganese, and poorer in alumina, than the Baltimore ores.

Wassaic Roasted Carbonate.

	Per cent.
Silica,	8.240
Peroxide of iron,	77.202
Alumina,	2.768
Protoxide of manganese,	3.005
Lime,	1.650
Magnesia,	1.167
Phosphoric acid,	.275
Sulphur,	.224
Loss by ignition,	5.684
Metallic iron,	54.042
Metallic manganese,	2.165
Phosphorus,	.120

Analysts: Booth, Garrett and Blair.

* *Trans.*, xii., 91.

† It is unfortunate for the sake of nomenclature that in the title Mr. Gridley called his metal "cast-iron." Since it had not been re-melted, it was not cast-iron but pig-iron.

‡ *Jour. U. S. Ass'n Charcoal Iron Workers*, vol. iv., p. 186.

§ *Ibid.*, vol. v., p. 45.

Although this ore has long been known this was the first and last time it was ever smelted. It had in some way acquired a bad reputation among the furnace-men, and no one would touch it. These tests, however, show that under proper conditions it will make a very superior pig-iron.

This strong iron is very different in composition from the Muirkirk, as is shown by the following analysis:

Wassaic Pig-Iron No. 4.

	Per cent.	
Graphite,	2.310	Total, 3.090
Combined carbon,	.780	
Silicon,	1.307	
Sulphur,	.086	
Phosphorus,	.294	
Manganese,	1.512	
Iron,	93.700	
	99.989	

Analyst: Dr. T. M. Drown.

The recently published results of Keep, Mabery and Vorce* on the influence of aluminum on cast-iron, taken in connection with the aluminous character of the Baltimore ores, might naturally lead one to suppose aluminum to be the cause of the strength of this pig-iron, but such is not the case. I have very carefully examined drillings from the other end of the pig from which the Thomas tests were cut, using a method which gives results easily within 0.001 per cent.; and while a slight precipitate of phosphate of alumina was, indeed, produced, it indicated less than 0.0001 per cent. of aluminum, which is less than the error of manipulation.

I have also examined the Wassaic pig-iron for aluminum with the same result, that less than 0.001 per cent. of aluminum was found. The specimen which gave the tensile strength of 41,882 pounds per square inch was used for the examination.

As to the cause of the unusual strength of these irons, the data in regard to the Wassaic iron are too meager to yield any satisfactory explanation alone; while, with all the data possible in regard to the Muirkirk iron, and with much consideration of the subject, extending over several years, about all I can say in explanation is a series of negatives. The secret is not in the composition of the ores; for in other furnaces the Baltimore ores have not yielded such strong

* *E. and M. Journal*, vol. xlvi., p. 125, and *Iron Age*, vol. xlii., p. 399.

iron, and the Wassaic ore is totally different in composition. It is not in the percentage of phosphorus, silicon and sulphur in the iron; for the Stickney No. 4 iron is practically the same in this respect as the Muirkirk, while the Wassaic is totally different.

The manganese varies greatly in the analyses of the Muirkirk pig, the highest in the strong irons (1.441 per cent.) being practically the same as the Wassaic (1.512 per cent.), while in the Thomas test (0.94 per cent.) it is practically the same as in the Stickney No. 4 (0.834 per cent.). Again, in the first Graff analyses of the Muirkirk strong iron, manganese was down to 0.565 per cent.

The total carbon is also somewhat erratic. While the Muirkirk-Thomas test (2.93 per cent.) and the Wassaic (3.090 per cent.) are practically the same, both the Graff analyses of Muirkirk are quite different, showing 2.705 and 3.730 per cent.

It may be that a certain balance is required between the constituents, without which the strong iron cannot be produced; but, given that balance, some other determining cause is evidently required to give the iron the extraordinary strength observed. The cause of the strength is not in the ordinary conditions of running the furnace, since they are the same for several furnaces in the region, which have never produced strong iron, as for the Muirkirk. It is evident that the cause is physical rather than chemical. Probably it is some unusual, but frequently occurring, condition of running which has not yet been discovered. The lines of the furnace have been materially changed several times without affecting this question.

The results of the recent grading of the pig at the furnace show that the physical characters of the metal are sufficiently pronounced to enable a practiced eye to discover the strong irons.

The fracture of the strong iron is surprisingly like some of Mr. Keep's tests of iron containing aluminum, the increased strength of which he ascribes to the carbon being forced into the graphitic state in minute scales, just as the iron solidifies. There may be the same separation of graphite in this case, which would explain the similarity of the fractures; but, aluminum being absent, there is yet no explanation why such a separation should occur.

The method of examining pig-iron for aluminum was developed in another investigation, which is not yet ready for publication; but I give an outline of it here, as it may be of interest on account of the attention given to the subject at this time.

The method of analysis is capable of giving results of any desired accuracy, since any amount, even a kilo, of material may be used.

With 25 grammes the results fall within 0.001 per cent. of aluminum. In this case 25 grammes were weighed into a capacious flask; 300 c.c. of H_2O were added, and to this 90 c.c. of pure concentrated HCl were added in three or four portions. The flask was provided with a valved stopper.

The next morning the solution was gently heated for one and one-half to two hours. The solution was then rapidly filtered into a second large flask, the residue boiled a short time with 5 c.c. of strong HCl, and the filtration was completed. All this work was done as rapidly as possible, and occasionally a lump of $HNaCO_3$ was dropped into the receiving-flask, in order to prevent as far as possible the oxidation of the iron.

The solution was cooled, and a regular basic acetate precipitation was made. Owing to the small amount of sesquioxide of iron, and to the SiO_2 present, it was not easy to hit the neutral point, neither was it necessary to do so exactly. If the Fe contains any considerable P (0.10 per cent.), this precipitate is a dirty green phosphate.

In order to catch any Al that did not come down in the first precipitation the filtrate was again brought to a boil, whereby, if the solution was not too acid, a slight precipitate was formed from the Fe that had oxidized during the filtering. If the solution was too acid a few drops of dilute ammonia were added.

The two precipitates were united, dissolved in HCl, and SiO_2 was separated.

With pig containing less than 1 per cent. of Si, these precipitates may be dissolved by putting them moist with the filter-paper into acid, but with more than 1 per cent. Si, the filtering from the filter-paper may become very tedious, owing to the SiO_2 dissolved. In this case it is better to dry the precipitates thoroughly, remove them from the paper, burn the latter at a low temperature, and add the ashes to the precipitates.

A second basic acetate precipitation, using ammonia and acetate of ammonia, was then made. If the work has been done carefully and expeditiously we have now all the Al from the 25 grammes of pig concentrated into less than 0.2 gramme of material.

The further treatment of this precipitate will depend upon circumstances. If much Al or little P_2O_5 is present it may be weighed, the Fe titrated, and the P_2O_5 removed by molybdate solution and determined, giving the Al by difference.

In this case, as in the case of most pig-iron, there will be so much P_2O_5 present that this precipitate will be very fusible, and unsuitable

for weighing. It is therefore, treated with pure NaOH (from metal), which separates all the Al and all or nearly all the P_2O_5 from the Fe. The solution is filtered, neutralized with acid, and the Al is precipitated by $(NH)_4OH$ as phosphate, which is ignited and weighed. Since the composition of this phosphate is not constant, the weighed precipitate must be dissolved in HNO_3, and the P_2O_5 separated and determined, giving the Al by difference.

All the work after the first basic acetate precipitation was done in platinum.

When large amounts of material are taken, it may be necessary to reduce the Fe before the second basic acetate precipitation.

Both of the pig-irons discussed in this paper are well represented in the collections of the National Museum: the Muirkirk, by a full series of the ores, fuel and flux, the product, including the pig and many of the test-pieces, and the slag; the Wassaic, by several samples of the ore and the product, including the pig and nearly all the test-pieces.

In conclusion, I wish to express my hearty thanks to Mr. Coffin for many courtesies in enabling me to make a thorough study of his furnace, and the strong iron produced.

NOTE UPON SOME RESULTS OF THE STORAGE OF WATER IN ARIZONA.

BY WILLIAM P. BLAKE, NEW HAVEN, CONN.

(New York Meeting, February, 1889.)

As the storage of water for agricultural and mining purposes in the arid regions of the West is now receiving much attention by the people and their representatives, some facts concerning the greatest pioneer enterprise for the purpose should appear on our records. I refer to the dam and reservoir at Walnut Grove, in Yavapai county, Arizona Territory, which was finished in the winter of 1887, and is known as the Walnut Grove Dam.

It is not my intention at this time to give a full description of

this work, but some general statements and approximate measurements will serve to give an idea of its extent.

The Hassayampah river, so famous in the annals of the Territory for its fitful moods of alternate floods and dryness, where at times an ocean steamer might be floated, and where at other seasons it is said that the fish must carry canteens, was the stream selected for this enterprise. When in flood it is one of the longest of the streams of Arizona. Rising in the granitic mountains near Prescott, it flows southward to the Gila, at Gila Bend, cutting through the ranges which separate the higher plains and large valleys from the Piedmont country sloping to Phœnix, Wickenburg, and the Gila Valley. The descent is rapid, and the floods from the summer and the winter rains rapidly run down the valley and are lost.

At the lower end of the somewhat open valley of Walnut Grove the stream enters a cañon of gneissose and granitic rocks, and this was the point chosen for the dam.

This dam is constructed of granitic rock, rough-laid without mortar, and is planked on the back- or water-side with two thicknesses of yellow pine plank, 3 x 8 inches, spiked upon vertical cypress and pine timbers. The outer layer of plank breaks joints with the inner layer, and the whole is caulked and painted with paraffine paint. The dam is 420 feet long on the top, 138 feet thick at the bottom, about 15 feet thick at the top, and 110 feet high in the center. It contains between 45,000 and 50,000 cubic feet of rock.

Before the construction was undertaken, several serious questions arose, which could be answered only by reason and faith, and not by experience, at least not by experience in our country in a region of like conditions.

One great question was, whether there would be sufficient water to fill the reservoir; another, whether the stoppage of the flow would cause the stream to dry up below the dam, and thus, by diminishing the flow, impair the water-rights, as, for example, the supply to the Vulture 80-stamp mill, taken out below the village of Wickenburg. The importance of the answers given by experience to these questions must be my excuse for bringing this subject before you.

The reservoir was filled by the first floods, and the water rose rapidly to and beyond the 80-foot contour-line, spreading over the farms up to the stakes set by the surveyors, floating fences and light wood, and approaching the doors of one of the oldest settlers, who

had defied the Apaches years before, but now ran pleading for the gates to be opened to save his adobe. Where before there was a broad sandy valley requiring irrigation, there is now a lake more than a mile in length, its waters stocked with carp, and the resort of wild fowl.

As to the effect upon the stream below, there has been an agreeable surprise. Either from a partial opening of one of the gates or a leak, there has been a constant flow of water from the dam; and this has kept a constant stream through the valley, giving more water than usual along its course, so that instead of the owners of water-privileges denouncing the dam and asking for injunctions, they are hoping that the dam will always leak to their advantage.

These results are of great value as a demonstration of what the function of such dams and reservoirs may be throughout the arid regions of the West. Even if not perfectly tight, they would be of immense value in catching the temporary floods, and in equalizing the flow of such intermittent streams as the Hassayampah and many others which might be named.

And I would impress upon the minds of the engineers and the residents of our arid regions, that there should be many such dams upon the same stream and upon the smaller streams and rivulets which at certain seasons have a considerable flow. Work should begin toward the sources of the streams, and small reservoirs should be multiplied in the mountains until the great bulk of the rainfall and the snow-water is held back and stored at the higher levels.

As an illustration of the importance of more than one large reservoir on the same stream, an experience below the Walnut Grove dam last summer may be cited. A heavy rain storm and shower—a "cloud burst"—swept over the mountains below the dam and suddenly flooded the cañon to such an extent that ditches and water-wheels were swept away.

This subject is a most important and interesting one, and falls within the domain of the labors of mining engineers. Most of the storage-reservoirs hitherto made have been for mining purposes, and now water is required for the reclamation of vast areas of the public domain, at present unfit for settlement, but often requiring only water and cultivation to make gardens in the wilderness.

THE COPPER-DEPOSITS OF COPPER BASIN, ARIZONA, AND THEIR ORIGIN.

BY WILLIAM P. BLAKE, NEW HAVEN, CONN.

(New York Meeting, February, 1889.)

COPPER BASIN in Yavapai county, Arizona Territory, about twenty miles southwest of Prescott, is well named. It is a depressed area, and a region of cupriferous impregnation.* The geologic conditions are simple. The foundation rock is a coarse-grained granite and gneiss, in which soda feldspar predominates. There are also dykes of porphyritic rock, and a large quartz vein containing pyrites.

Superimposed on this crystalline foundation, we find heavy beds of mechanically formed rocks, conglomerates, breccias, and sandstones, in horizontal layers cropping along the bed of a creek, and apparently the remnants of a much more extended formation, now denuded and largely carried away by gradual atmospheric erosion.

The heavy beds of conglomerate are in many places much broken and tilted up, even standing on edge in large blocks, as if they had been lifted by some great convulsion; but the cause is much more simple, being merely the removal by gradual disintegration and erosion of the softened and decayed granitic rock below.

The materials of these sedimentary beds are chiefly fragments of granite, gneiss, and plutonic rocks loosely mingled. They are the chief repositories of the copper-ore, which forms the cementing substance. This copper-ore is the blue and the green carbonate, azurite and malachite, and the ore is so generally spread through the mass of the beds that the blue and the green croppings can be seen at a great distance, particularly after or during a shower of rain, when the colors are extremely brilliant and beautiful.

The copper carbonate is not only a matrix and a cementing material for the fragments of rock, but it invests and covers these fragments so that only malachite and azurite are visible. The beds are from three to ten feet or more in thickness, and, although seemingly solid carbonate of copper, rarely contain over 12 or 15 per cent. of that metal. Some portions of the conglomerates are much more highly charged with copper than others, and in places the beds are almost without copper. If a lump of the coarse sandstone

* The principal deposits cover an area of about forty acres.

charged with the copper-ore is digested in acid, the carbonate of copper is rapidly dissolved out, and the mass falls asunder in loose grains of sand.

In the bed of the little dry "wash" there are many boulders, from a few inches in diameter to one or two feet, which are completely covered with a crust of azurite. If there are cracks and cavities in the boulders, the copper carbonate will be found inside or below the surface, but it is mostly upon the surface as a thin covering.

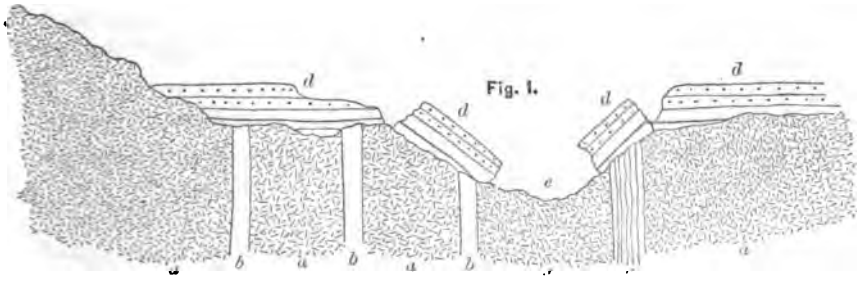
These copper-depositions are clearly the result of the gradual percolation of copper solutions passing through the porous sand-rock; and the copper carbonate is a deposit of incrustation, not of replacement, for, so far as the sandstones and the conglomerates have influenced the deposition, the action appears to have been mechanical rather than chemical. The surface, rather than the chemical composition of the strata, appears to have determined the deposition. Nor does it appear that the copper carbonate has replaced any calcareous or siliceous cement. The absence of a cementing material seems to have favored the infiltration and distribution of the cupriferous solution, which may have been gradually concentrated by evaporation on the surface of the coarse grains of rock. Assuming, what is very probable, that the copper was carried in as a dilute sulphate, it would be thrown down as carbonate on meeting carbonated waters, or meeting water or moisture in the rocks more or less charged with carbonate of soda. Sulphate of soda would be formed and flow away, to be perhaps concentrated by evaporation at some distant point, and form beds of Thenardite, a mineral which is abundant in Arizona.

These deposits of copper carbonate, covering, as has been observed, about forty acres, contain in the aggregate a very large amount of copper. The question of the source of such extensive depositions of copper at once arises.

The granite below the cupriferous beds, and throughout the copper area, is very much decomposed and softened, so that it may be cut with a pick or even a shovel. Considerable areas of granite surface are exposed, and are not covered by sandstone. There are numerous veinlets and thin seams of red oxide of copper, accompanied with malachite, and malachite also occurs disseminated in small nodular or concretionary masses, not much larger than kernels of corn, in the soft ferruginous clay resulting from the decay of the granite. These little button-like discs of malachite are so abundant

that they could be washed out with profit if water could be carried upon the ground.

There are also considerable quantities of red oxide of copper in thin sheets with malachite on each side. These are chiefly broken out from numerous small veinlets traversing the granite. The red oxide is in the middle and the malachite on each side next to the



Ideal Section of Copper Basin, Arizona, Grouping the Geologic Conditions without Regard to Distances or Scale.

a,a, granite; *b,b*, dykes; *c*, quartz; *d,d*, sandstones, carrying copper.

granite walls, the structure being as shown in Fig. 2, which represents the characteristic form of the small infiltrated sheets of ore seldom more than an inch in thickness, and commonly not over half an inch thick. The central portion is sometimes sulphide, not oxide. In some places fragments of such veinlets are very numerous, having been weathered out by the washing away of the softened granite, but no extensive or heavy deposits have been found to justify sinking or mining upon. Several drifts and tunnels have, however, been run into the face of the granite hill following some of the veinlets of copper-ore, but the results have not been satisfactory.

Some of the small veins consist partially of quartz, this being the veinstone, impregnated with malachite. These appear to have originated since the softening of the granite, and are probably the result of its partial decomposition.

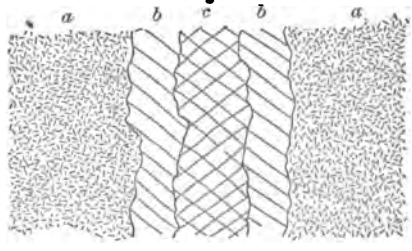
In one of the principal tunnels which has passed through the decayed portion of the granite and penetrated the unchanged interior portions, sulphurets of iron and of copper are found in small bunches and seams irregularly distributed. The quantity is not large, not enough to justify mining, yet in the aggregate the amount of cupriferous sulphide is probably sufficient to be the source of the oxidized superficial ores. Further evidence of the very general impregna-

tion of the granite with copper sulphide was found in the shaft sunk near the bed of the creek to obtain water. Yellow copper-ore is found there in small but solid masses in the midst of the hard gray unchanged granite.

I do not attempt an explanation of the source or origin of the chalcopyrite in the granite. Any satisfactory explanation would have a wide application. There is no evident relation of the sulphides to the porphyry dykes, nor, so far as observed, is there any pyrites in these plutonic rocks. This is, however, a matter for investigation.*

All the observed phenomena point to the disseminated yellow copper-ore as the source of the superficial copper deposits, and as

Fig. 2.



Section of Veinlets in Granite.
a,a, soft granite; *b,b*, malachite; *c*, red oxide.

the principal factor in the alteration of the granite. It would appear that the soluble sulphates of iron and copper, formed by the oxidation of the copper pyrites, and carried by endosmotic percolation through the substance of the rock, have acted upon the feldspars, promoting their decomposition with the separation of silica and the alkalies; the deposition of copper as oxide and carbonate; and the formation of alums, and probably also of sulphate of soda, which, flowing away, may have concentrated beyond. The formation of the extensive deposits of copper carbonates may also have been brought about by carbonated surface-waters, or waters more or less impregnated, as is not uncommon in those dry regions, with carbonate of soda. That soluble silica is one of the products we know, from the fact that in a little trickling stream of water flowing from a spring in the decayed granite there was an abundant separation

* As an example of copper in eruptive rocks, Daubrée cites the melaphyre of Norheim near Creutznach, which, according to M. Laspeyres, contains 0.00118 of its weight of copper.—*Les Eaux Souterraines*, ii., p. 134.

of gelatinous silica, colored green by copper. In this abundance of silica separated from the granite, we find an explanation of the origin of many of the little veins of quartz carrying copper-ore, and another fact in evidence of the origin of quartz veins generally.

Thus the granite becomes softened in place and "rotted" as deeply as surface-waters and air can penetrate; the sulphides disappear; silica and the alkalies are removed, and a ferruginous clay-like mass is left with disseminated accretions of copper carbonate and the formation of quartz veins.

An extended chemical investigation is necessary to determine all the conditions which exist, before the exact order of interchange of elements can be stated with confidence. These Copper Basin deposits of copper-ore in sandstone and conglomerate are very different from those found in the Triassic shales of New Jersey, and of Germany, and other countries where the ore is disseminated in grains or patches, and in nodular masses more or less isolated one from the other, and in the mass of the rock. In the Copper Basin deposits, the ore is spread continuously through the mass, and invests the grains of rock, forming a coating. The ore does not in any case appear to have been deposited by reason of the presence of any organic remains, as, for example, the stems of plants such as are found in the sandstones of northern Texas, or in Russia. Murchison has described cupriferous grits, sandstones, and shale, developed at the Zavods of Yugofski and Matovilika, where the ores of copper, chiefly the green carbonate, are disseminated, and are in the form of cupriferous concretions, generally around the carbonized stems of plants.* Not only Murchison, but Maynard, has described the deposition of copper on organic remains in the copper sandstone of the Urals.† These Arizona beds differ from the cupriferous Permian strata, not only in the form of the copper deposition, but in their much more modern and local nature.

The copper-deposits of northern Texas are familiar examples of the deposition of copper-ore on the remains of plants.

Wendt has mentioned the copper of Copper Basin as carbonate-ore which "must be of secondary origin,"‡ but otherwise it has not been noticed in our *Transactions*.

He considers the red oxide of the Santa Rita copper mines of New Mexico as derived from the native copper by alteration.§

* Russia in Europe and the Urals, p. 144.

† *Trans.*, ix. 33.

‡ *Trans.*, xv. 74.

§ "The Copper-ores of the Southwest," *Trans.*, xv. 27.

The explanation of the origin of the Copper Basin ores herein given will apply particularly to the conglomerate deposits of the Gila and Lower San Pedro, and also to any ore-deposits accompanied by extensive kaolinization of the crystalline rocks.

This Copper Basin ore is not a "smelting ore." The bulk of its substance is not sufficiently fusible. Its origin by infiltration suggests the nature of the process by which the copper should be taken out. It is an ore especially well adapted to leaching. The absence of soluble minerals other than the copper carbonate is a great advantage, while the open granular condition of the rock permits the ready action of a solvent and of subsequent lixiviation.

The conditions at once suggested the following process, which is given in outline merely :

1. The solution of the copper carbonate by dilute sulphuric acid.
2. The extraction of the copper by electrolysis.

The conditions favoring this method are numerous, but the greatest is the abundance in the region of auriferous pyrites, the "rebellious sulphurets," for which the gold miners are seeking the cheapest and best method of working. The distance over which these sulphurets must now be transported entails a heavy expenditure. The establishment of sulphuric acid works at Copper Basin would make a home market for the sulphurets, and while the sulphur is utilized for making acid the oxidized residues holding the gold could be chlorinated. Thus the sulphide ores on the one hand, and the clean carbonates of copper on the other, although difficult to utilize separately, would by combination give us the precious metals, and the copper in a marketable form at a comparatively low cost, and the problem of working both is solved.

The spent residue of iron-oxide might also be utilized (provided there was rail transportation) at lead smelting works.

There are several economies by such a process which will suggest themselves. No great supply of water is required.

The acid need not be concentrated: "chamber acid" would be strong enough.

It is possible, also, that no great quantity of acid would be needed, for when the bath is once formed, it is probable that the copper-ore itself may be made the anode, and yield its copper to the solution direct.

Pine wood for fuel is abundant on the adjoining mountains, and dynamos can be run very cheaply.

Our fellow-member, Prof. G. W. Maynard, some years ago, put

up a plant in Siberia for the treatment of copper carbonate conglomerate ore by lixiviation with sulphuric acid, and has promised to contribute to the *Transactions* of the Institute an account of the plan and its results in practice.

SOME THOUGHTS RELATING TO THE AMERICAN INSTITUTE OF MINING ENGINEERS AND ITS MISSION.

BY WILLIAM B. POTTER, E.M., ST. LOUIS, MO.

(Presidential Address at the New York Meeting, February, 1889.)

IT is a time-honored custom in this, as in other kindred bodies, for the retiring President on giving place to his successor, after a year of official duties which have been the means of directing his attention in a more than usual degree to the work and interests of the Association, to report the results of his observations from this outlook, and the thoughts that may be suggested by the view.

In looking out over the many fields of activity within the range of this Institute, the attention may be arrested by special features marking recent progress and development, or such as by their meager outline or lack of important detail stand in less pleasing contrast, or even form unsightly objects in the view. It may be that the special field and particular work in which one is engaged will, when observed from the more distant standpoint, or in the light of the whole outlook of the profession, assume a different aspect or develop features of new interest and importance. In this way some special object may suggest to the observer a fitting theme for the occasion.

To others, again, an outline of the whole view may seem a more useful subject for study and review. All fields are examined and the wide horizon of the whole profession scanned to note and report whatever is new in knowledge and practice—the new permanent bench-marks that appear near the horizon designating ground definitely located; the stakes which mark reconnoissance into new territory, and even the prominent objects that faintly appear through the mist of the distant atmosphere, and which may serve later on as points of attachment for the advancing lines of survey into the unknown. And besides these, there is the progress of the work in older and nearer fields—what has been attempted, what accomplished,

as well as the means by which success has been achieved or failure brought about.

But the outlook has become so extensive and the work so specialized and developed in detail, that, even if it were within my power to present on this occasion a complete view of the whole field in any way worthy of the scene, your time and patience would be exhausted; while any rough sketch outlining the tendency or progress of development must necessarily be too meager of detail and offer too little contrast of light and shade to form a satisfactory representation of the whole scene for any purpose.

The task which I have undertaken, although an unusual, is a more modest one. It is to present briefly a few thoughts that have suggested themselves during the year, relating to our own Association—the American Institute of Mining Engineers.

The Institute is now fast approaching the year of its majority. More than eighteen years ago it came into existence in the town of Wilkesbarre, Pa., and was welcomed as a long-desired heir, who should, in time, gather together and organize the scattered and neglected interests of the great estate of the profession, and be the means of adding new riches of knowledge to the store and new honor to the name, as well as of benefiting mankind. The achievements of all these years have proved that the expectations were not in vain. The twenty-two who first assembled at Wilkesbarre to organize the Institute are, many of them, still with us; but their number has increased, until now there are eighteen hundred or more. All parts of this continent are represented in the membership, as are the chief countries of Europe and South America and parts of Asia, Africa, and Australia; and the list embraces the names of many who have been foremost in the achievements of all branches of the profession. The fifty-two meetings that have been held in the leading industrial and commercial centers of the land have been the means of extending the influence and reputation of our calling and promoting the interests of true science and sound practice. Not far from a thousand papers have been contributed upon subjects relating to the constitution and occurrence of Nature's varied stores, as well as their exploitation, preparation and treatment to adapt them to the use and for the benefit of mankind. The sixteen volumes of published *Transactions* containing this liberal contribution form a rich and useful store, and furnish abundant proof of ability and zeal as well as faithfulness of members to the cause the Institute upholds.

The uncertainties of infancy and childhood have thus been safely passed, and the growth and strength of vigorous youth have already been acquired. Even the earnestness of purpose and judgment which betoken a ripening manhood are not wanting in this case. But with a growth that has been so rapid and of such vast extent, is there not a possible danger that such a body may become too clumsy and unwieldy; that for lack of broader training, and by not exercising all its powers, it may fall short of complete development and fail to reach that sphere of usefulness such an organization should occupy?

While looking back with satisfaction upon the eighteen years that have passed, and congratulating ourselves upon the prosperity and excellent standing the Institute now enjoys, it may be well, in view of the dangers that come with continued growth and the responsibilities of advancing years, to consider for a little the mission and opportunities of such a body, and the duties which these impose.

The general plan upon which the Institute is founded and its affairs are administered has proved a wise and efficient one. Avoiding exclusiveness and narrow limitations as to membership, a generous welcome to associate membership is here accorded to all persons who may fairly be considered suitable and are desirous of joining the Institute. A large and useful constituency has thus been developed, representing all classes and shades of interest in matters with which our organization has to do. The administration of its affairs being delegated by the whole body to the selected few composing the Council, responsibilities are concentrated, and many evils which might result from so large and scattered a membership are easily overcome. The rashness and ignorance of individuals can here do but little harm, while ability and merit find encouragement and fields in which to win rewards. Each member will find his level, whatever his proclivities may be.

The three meetings held each year in various parts of the country serve in a large measure to keep up the interest of scattered members, and to prevent many of the evils due to too great a localization of the strength and activity of the Association. The liberal supply of papers and the facilities afforded for their early appearance in convenient form do much, also, to impress even the most distant member with the value of his relations to the Institute.

But, in spite of these advantages, there is evidently a danger that with so large a membership and so great a diversity of special interests the work may become somewhat scattered and diffuse; that

the more serious business may be too lightly passed over, and important means for advancing the welfare and usefulness of the Institute be too easily neglected for an undue development of the popular features. Already there is a noticeable tendency at the several meetings of the year to magnify the importance of excursion and entertainment, and use the working sessions mainly to fill up the necessary gaps in the programme of picnics, as if to afford needed rest and relaxation to the exhausted Local Committee as well as visiting members. There can be no question as to the value of the excursions, or even the entertainments, if kept within their proper spheres; but surely these should form subordinate features, and constitute rather the pleasing accompaniments to the more serious and solid business of the meeting. The sessions will otherwise become but little more than occasions for the announcements of local committees and the reading of papers by title, the formal election of a batch of new members, and the hurried adoption of the usual vote of thanks.

Special gifts of prophecy are not needed to foretell the future if we thus permit the Institute to drift along. It will become an overgrown and weakened body with all its old spirit and earnest purpose gone. Instead of invigorating its members by healthy exercise of all its powers, and creating an active stirring interest in the several meetings of the year, there will be but one bond of union to hold the scattered parts—the occasional volume, containing papers exploited from good-natured and unwary members by a skilled official, who will have learned how to apply all the experience of the profession in prospecting for contributions and engineering them into print. For the vigorous, living Institute, which is capable, by keen but friendly criticism in open discussion and debate, of spurring on the willing member to investigation and research, and encouraging all to voluntary contribution of useful experience or of important facts, we should have but an organized bureau of publication, offering as a bonus a week of cheap excursions at stated intervals through the year.

No one will question the value of the published *Transactions*, or the importance of keeping them up to the standard already attained; and the right to criticise and discuss any paper by written communication is fully recognized. But we all know that it is the interest given by personal presence and the zest of face-to-face discussion and debate, if properly prepared, that best draws out the experience and judgment of individuals, and leads to broader views and fuller

knowledge as well as further investigation and more careful thought. By preparing for and encouraging full discussions at the meetings and having them published, under proper revision, with the papers to which they pertain, a greater value would be given the publications than even at present obtains. Such discussions often contain an amount of pith and point not always found in the papers which excite them, and their omission often seriously detracts from the completeness of treatment which the subjects might receive. The meetings would surely prove more attractive, especially to those members who, by their work and interest, can contribute so largely to the extension of the influence and usefulness of the Institute; and even the most distant members, and such as by occupation are prevented from being present at these meetings, would be benefited thereby.

It is hardly to be expected, perhaps, in an organization grown to such proportions as the Institute has assumed in the number of members and variety of interests represented, that a very efficient discharge of all its duties could be accomplished without the assistance of the systematic methods which a more definite organization would supply. In the scientific associations of wide and general range, the several interests are usually classified into groups and sections more or less fully organized and equipped for independent work; and it has already been suggested that it might be well for the Institute to adopt a similar course. But while we have here a sufficient variety of interests to justify such a development of working-sections, there are some objections that might be urged. It would require important amendments to the present rules to secure what might prove by the experiment a failure; and it would seem possible to obtain all the valuable features of such a system by a simpler process requiring less formal action. With a complete organization of independent sections there might be very unequal development of some parts, causing more or less jealousy and friction, and tending to give an undue bias to the Institute at large. It is probable that the strength and influence of the organization would thus be weakened, and result possibly in the separation of important parts.

All the advantages of a system of organized sections might be secured, and the many evils avoided or controlled by the use of means already at hand, without requiring any change in the present rules or organization. A special committee of three or five might be appointed to represent each group of related interests in the Institute, and be charged with the duty of arranging the details of all

business connected with their respective groups. Such business would consist mainly of arranging in advance for papers to be read and subjects to be discussed relating to their groups at the several meetings of the year; promoting so far as may be the attendance of such persons as would have a special interest in the subjects, or be likely to add to the interest of discussions; looking after the details of excursions and visits to places and objects connected with their groups; providing definite information concerning plant and process, materials and products, and other matters of interest, as well as seeing that the crowd of visitors is properly organized into parties of inspection, with intelligent guides to give the needed explanations and information. These are some of the more apparent duties that would devolve upon such committees; but many others would doubtless be developed in the exercise of their legitimate functions, and in still other ways contribute to the interest of meetings and the general welfare of the Institute. Efficient aid could by such means be extended to local committees in the preparations for meetings and the successful carrying out of the programme provided. There could never be any uncertainty as to the success of a meeting or embarrassment of riches in the way of entertainment; and we should be spared the not unusual spectacle of a train-load of people turned loose upon some establishment to wander about aimlessly, without philosopher, guide, or friend, getting little or no definite information, and not adding in any way to the picturesqueness of the scene. Thus, without an elaborate organization, or the use of unusual means, all the real advantages of the fully-developed section could be realized; and, with the controlling influence of the Council to harmonize all the parts, there need be no abnormal development or bias, and nothing to disturb existing relations or cause embarrassment of any kind. The action required to secure this desirable result is of so simple and informal a nature that it could be applied at any time, and the scheme could as readily be given up whenever it proved unsatisfactory. There is nothing in our present rules to interfere; and, indeed, such a measure would seem rather a means for more effectively accomplishing the objects specified in the first article as those for which the Institute was founded.

In the classification of interests under such a plan, there should be little difficulty experienced, if it is made sufficiently general at first, and capable of embracing all subjects with which the Institute has to do. As a suggestion for such a grouping at the start, the fol-

lowing might serve: I. Iron and Steel; II. The Precious and Base Metals; III. Geology and Mining; IV. Chemistry. Such a classification is simple and natural, and may be made to include all subjects within the range of the Institute. Iron and steel would, of course, form a very important group of related interests. The precious and base metals are so closely associated in the character and metallurgical treatment of their ores, as to make it difficult to classify them in separate groups, and the whole would include a series of interests sufficiently well represented in the Institute to form a strong and valuable group. Subjects connected with geology and mining would naturally be classified together, since the occurrence of ores and mineral deposits and the character of the formations with which these have to do have an important bearing upon the problems connected with mining. With such an important range of subjects, and so strong a following in the Institute, this group should develop marked activity and interest. While chemistry plays an important part in all the other groups, it has so many special features and a province so peculiarly its own as to call for a separate group. It would constitute, also, a very important and useful group; since there is a large number of specialists in this line within the Institute, and the need for growth and development in the field of activity in which they labor is very pressing.

Numerically considered, the interest in these groups throughout the membership would not be unfairly divided; and in geographical distribution, symmetry would not be lacking. The interest in iron and steel occurs mainly from the Mississippi valley eastward; while that in the precious and base metals starts in the same interior region and sweeps, for the most part, to the westward. In the other groups, all parts of the country have a nearly equal interest, with a slight preponderance in favor of chemistry in the east, and geology and mining in the west.

But, it may be asked, will the Institute, in doing this work, even in a complete and efficient manner, be doing its whole duty to the profession and the cause in which we are engaged? With an association of eighteen hundred members or more, scattered over the whole country, and engaged in all branches of the engineering profession, and in the multitude of occupations connected with the mining and treatment of minerals, and the utilization of the wonderful variety of products obtained from these—with such a band of active workers in field and mill, in laboratory and office, can it be said that all the opportunities afforded in the Institute for advancing the in-

terests of our calling are taken advantage of by meeting thrice a year for the reading and discussion of papers, and publishing the results? The first article of our rules specifies that "the objects of the American Institute of Mining Engineers are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications, among its members and associates the information thus obtained." From a literal interpretation of this it would seem at first that all the means provided for meeting the objects specified were already employed, and yet it cannot be argued that the manner in which this work is to be done is restricted or prescribed. And, further, the view to be taken of the mission of this Institute, after eighteen years of wonderful growth and prosperity, and progress in the arts and sciences with which we have to do, is surely much more extended than when the twenty-two pioneers met at Wilkesbarre and specified the objects of the organization they were about to found.

The manner in which the Institute has been doing its work all these years has contributed in a remarkable degree to the promotion of "the arts and sciences connected with the useful minerals and metals, and the welfare of those employed in these industries." A vast amount of knowledge has been accumulated and placed in such a form as to constitute a rich and convenient store-house of reference concerning the great variety of subjects connected with our calling. And far more, even, than this has been contributed and remains unrecorded in our annals, except in some of the final results—a development of practical experience in details of skill and keen technical instinct, all which should prove a lever of vast power if rightly directed, for the working out of the problems now before us and such as the progress of the future may bring.

But with all this variety of knowledge, and with the skill and experience that have come with it, is it not the duty of such an association as this, and, indeed, of the three great engineering associations of the country—the Civil and Mechanical Engineers as well as our own Institute—to do something towards reducing this knowledge and experience to a more systematic and definite basis; to determine how far the existing but scattered knowledge of law and fact will suffice; to provide for securing by test, experiment and research, important data that may be lacking, and bring to bear all the wisdom

and judgment that may be required to reach conclusions that are definite, and at the same time wise and just?

I am not unaware of what has already been done in this direction, or of the good that has resulted. But when we consider the vast amount that remains to be done, and the importance of doing it, how insignificant seems the little that has been accomplished. The Institute, too, with all its opportunities, has contributed but a small part of this. Some committees have been appointed for the consideration of such work; but they have mainly died reporting progress, or after returning little more than a preliminary report. Some individuals have contributed valuable work in this direction; and this is well so far as it goes. But what we need now are the persistent and best efforts of the Institute and of the other engineering associations, separately or conjointly, to prepare for and, so far as may be, secure the adoption of uniform standards. Surely, there is no engineer in any branch of the profession who has gone outside the limits of his own closet, but appreciates the evils of the existing state of things. There is hardly time on this occasion, nor should it be necessary, to discuss at length any of the many cases calling for treatment in this connection; but it may be well to refer briefly to a few of the more prominent by way of illustration.

The testing of engineering materials is a subject which only needs to be mentioned to prove the necessity of reforms. The results of tests of the various materials used in construction are to be found in great numbers; yet in the majority of instances they are not only in a large degree unreliable, but often misleading. In some cases, not even the shape, size, or proportions of the test-pieces used are given; more often, important details relating to the manner of applying the test are missing; while in the vast majority of cases no thought whatever seems to have been given to the mechanical treatment to which the test-piece has been previously subjected; and yet the results often depend as much upon this as upon any other feature in the case.

Standard specifications for various classes of material under special applications is surely a subject of sufficient importance to merit the attention of this and the other engineering societies. One need only point to the many wild and useless specifications for cast-iron in which tensile and compression-tests are minutely provided for, and all important tests for resilience are wholly ignored. The paper read at the last meeting of the Institute on "Steel Rails and Specifications for their Manufacture" is an evidence of the value of such

work and its utility for all concerned. Considering the manifold conditions of practice and application, such specifications cannot always be drawn in complete detail, but they would serve as good types and useful guides and raise the standard of excellence throughout.

The subject of boiler-trials and the steaming-test of fuels furnishes another important field. Of the many trials reported, but few will stand the test of critical examination. Some of the necessary data are likely to be missing; and usually it is the all-important analysis, showing the quality and condition of the coal used and of the ashes and chimney-gases.

The want of a uniform standard of gauges and a system of sizes for machine-shop and factory gives rise to an amount of annoyance and useless waste that should impel us to better things.

What excellent results might be obtained from a careful study and comparison of the various methods of sampling and chemical analysis, and the adoption, so far as may be, of standard methods for each. Greater accuracy and usefulness would be secured. Personal equation, which is now so unmanageable in these matters, would be reduced to narrower and far more definite limits. Is it not time, too, to develop a reliable, practical system of proximate chemical analysis of the metals in their many forms? The ultimate analysis of organic substances is of little value in the determination of their properties and applications. Some form of proximate analysis is relied upon to give us a knowledge of these. We have daily proof of the relation of the properties of metals to the proximate combinations in which the ultimate elements are arranged; and yet we have practically nothing but the results of *ultimate* analysis to depend upon. Already the limits of usefulness have been reached in the use of ultimate analysis for determining the properties of metals and inorganic substances.

With these and many other lines of inquiry pressing hard upon us, the Institute should be awake and active, contributing its share, not only of labor, experience, and skill, but, whenever needed, its funds as well. With an accumulated fund of about $15,000 and an annual surplus of $4000 or more, and with the ability and experience in its membership, this organization is surely well equipped for the more aggressive work that is needed. By the appointment of special commissions, and the judicious appropriation of funds for investigation and research, there should be no lack of important results.

NAILS FROM TIN-SCRAP.

BY OBERLIN SMITH, BRIDGETON, N. J.

(New York Meeting, February, 1889.)

IT may surprise the learned metallurgists who read this paper to learn that, by a recent discovery, nails of good quality can be made at one operation, directly from the ore, at the rate of, say, sixty per minute for each operator. It should be stated, however, that the raw material referred to does not answer strictly the ordinary assayer's definition of an ore. It is found, in strata of various thickness, in Harlem, N. Y., and other localities where the *débris* from restaurants and from sheet-metal factories of various kinds has been dumped. In other words, it is old and new tin-scrap—one of the few substances which this generation, mainly occupied in exhausting the accumulated resources of the past, seems to have laid up, by way of atonement, for the benefit of posterity. In speaking of this material as *ore*, we are simply looking forward, prophetically, to the time when our descendants may dig it up and write learned papers for the American Institute of Mining Engineers upon the best methods of assaying and smelting it.

At least such has seemed to be its destiny hitherto. It may fairly be said that the many attempts which have been made to utilize it by separating, through chemical or electrolytical processes, its two valuable constituents, metallic tin and first-class wrought-iron, have failed, either technically or commercially. The reasons for such failure need not here be discussed. Either the separation has been incomplete, the iron still retaining enough tin to spoil it for the sinking-fire, or other use short of re-melting (perhaps even for that), or the manipulations of the process have been too expensive to make its results profitable—or both. Meanwhile the great tin-scrap deposits have gone on growing faster than any other strata of our Post-Tertiary, Psychical Era; given up by metallurgists, not yet attacked by geologists, and explored only by that mining engineer of the transitional period, *Gulielmus Caper*.

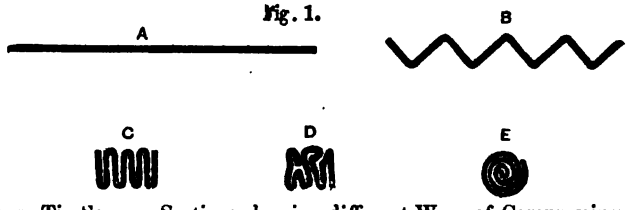
Before leaving these heaps of tin-scrap, however, to become mere mineral deposits for future ages, it may be well to consider a novel plan for their immediate utilization—novel, not only in its means, but in its principle. For it undertakes to use this material just as it is, without trying to separate its constituents at all, and to use it,

moreover, for a purpose in which the qualities of both these constituents, namely, the tensile strength and ductility of the iron and the resistance of the tin to corrosion, are directly employed with advantage. I refer to the manufacture, by mechanical pressure, of nails.

This nail was invented, in its original shape, by Mr. Geo. H. Perkins, of Philadelphia, and has been developed, through various forms, until it has almost reached a commercial stage—the machine in which it is to be made, in marketable shape, being nearly completed. The writer has been associated with Mr. Perkins in its development, and can, perhaps, make interesting a brief description of the troubles we have gone through in order to produce it quickly, at one operation, in a reasonably manageable and durable machine.

The first scheme tried was to cut "blanks" from ordinary tin-plate and sheet-iron scrap, of an approximately rectangular form,

Fig. 1.



Nails from Tin-Scrap. Sections showing different Ways of Compressing Blank.

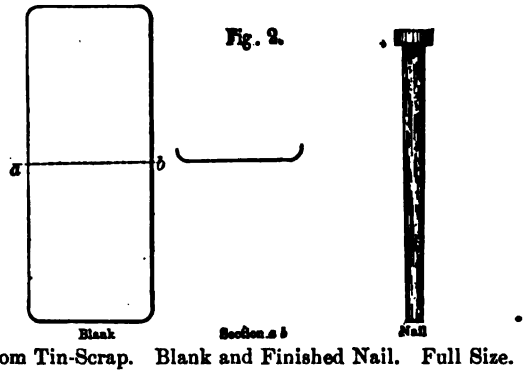
about three-quarter inch wide by one and three-quarter inches long, similar to the plan shown in actual size, Fig. 2, but having the end-section A, Fig. 1 (drawn about twice its real size). This was done in an ordinary press. By a second operation, it was corrugated into the form B, Fig. 1, in another press, with a special automatic die, which corrugated the middle grooves first and the side grooves afterwards; much of the metal used not proving strong enough to stand the friction of being pulled into the corrugations of the die without cracking. At a third operation, the embryo nail was crushed together into the form C, Fig. 1 (but more tightly closed), something after the manner of closing an accordion bellows. These headless nails were then fed by hand into a revolving dial, to carry them under a heading mechanism, after the operation of which they were ejected from the machine automatically.

Obviously, this series of separate operations made the nails far too expensive for the market; and, moreover, some trouble occurred from their splitting in the tightly folded corrugations. We afterwards built an *automatic* machine which performed all the above

operations in succession, delivering a complete nail at each stroke. It was, however, far too complicated for practical use, and simply served to show what could *not* be done.

Our second nail-machine proper, constructed to receive blanks, which had been already cut in an ordinary press, worked fairly well. It was likewise an experimental machine, built "piece-meal" and not strong enough for continual hard work; but it served this time to show what *could* be done. Instead of working upon the principle of regular corrugations, it simply crushed up the blanks edgewise into any form which they chose to assume—the end sections appearing somewhat as shown at D, Fig. 1. The blank and finished nail are shown in Fig. 2. Sometimes the blank is made tapering.

The machine now under construction has been very much simplified and made enormously strong and heavy. It is adapted to cut-



Nails from Tin-Scrap. Blank and Finished Nail. Full Size.

ting, crushing, griping and heading the nails at one operation, and can be run as fast as an expert operator can feed the material. Its feed probably varies, with jagged, irregular scrap, from thirty to ninety nails per minute, although straight strips of sheet-metal can easily be fed by hand into a machine running as high as 240 strokes per minute.

During the course of our experiments, various forms of nails have been tried. Among others were straight cylindrical nails with conical points; straight, square nails with pyramidal and with wedge-shaped points; hexagonal nails, etc. The most practical form, however, seems to be the square taper nail shown in Fig. 2, which has about the same shape as the ordinary cut nail, but is somewhat stronger and a good deal tougher. It is well-adapted for all ordinary purposes, but is especially suitable for a roofing nail, since the tin coating prevents much rusting, and is good to solder to.

Among other processes, we have wound the nail blank upon itself, after the manner of a window shade, but minus a mandrel. This makes a very neat nail.

The economy of this system of nail-making is obvious. The scrap can be bought for about seventeen cents per 100 lbs., and a boy can make from 80 to 100 lbs. of nails per day. The most economical system of manufacture will probably be to run one or more nail-machines at each large "tin-shop," set as close as possible to the presses which produce the scrap, so as to avoid the expense of unnecessary handling, and the extra tangling-up incident thereto.

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*GOLD-MILLING IN THE BLACK HILLS.**

BY H. O. HOFMAN, RAPID CITY, DAKOTA.

(New York Meeting, February, 1889.)

WITH the exception of the exhaustive paper on the Father de Smet mill, by its designer, Mr. A. J. Bowie, Jr. (*Trans.*, x. 87), nothing, so far as the writer is aware, has as yet appeared on the stamp-mills of the Black Hills of Dakota, although they have been steady producers for the last ten years. Within the small area of about 6000 by 1600 feet, $2,271,341.14 was produced in 1887† from rock averaging $4 per ton in free gold.

I. THE ORES.

Character of the Ores.—The gold of the district is found in quartz and pyrites finely distributed through vast bodies of mica- and amphibole-schists, argillites and phyllites, and also impregnating the schists themselves. The "Belt," embracing the section of Lead City, Terraville and Central City, is the only part worked at present.

The principal associated mineral is iron pyrites, with some arsenopyrite and chalcopyrite, garnet and asbestos.

The ores from open cuts and upper levels are more free-milling than those from underground workings below water-line. Hence

* The substance of this paper, with some additional matter, will be published, by permission of the Council, as a part of the Annual Report of the Dakota School of Mines, with which Prof. Hofman is connected.

† See Report of the Director of the Mint for 1887.

the mills running on oxidized ore have tailings as low as $0.25 per ton, while tailings from unaltered ore run sometimes up to $2.25 per ton. By watching the pulp when it flows down the plates it can be readily determined whether the ore comes from higher or lower levels. In the former case it will generally have a brownish red color, in the latter a bluish gray. The amount of free gold in the ores varies, therefore, with the depth at which they are mined. It is difficult to give a correct average figure of its value, but $4 per ton will be near it.

The total value of the ore is not definitely known, as its weight is not actually determined, but only estimated, and regular dry assays are not made at all.

To determine the amount of free gold in the ore, the following method is in practice: Samples are taken daily from the different workings in the mine and sent to the "sampler," who crushes them, pans them and estimates the gold in the pan. Every valuation thus made is booked, and at the end of the month the average is taken and compared with the output of the mill, and the amount of gold recovered thus approximately determined. The mode of operation practiced by the "sampler" is simple: The sample, weighing, say, ten pounds, is emptied into a 4-gallon bell-shaped mortar (13½ by 12½ inches) and crushed to nut-size with a heavy pestle, swung from a spring-pole. From it two pounds are then transferred into a second mortar of the same size, with a wooden lid, and there pulverized wet to a fine pulp, by means of a small steam-stamp, which is in reality an old power-drill, fitted up for this purpose. When sufficiently fine (as judged by the ring of the pounding stamp), the pulp is panned until all pyrites and other heavy sands are washed off with the tailings, and only the free gold remains. The "sampler" of the Homestake company pans from fifty to fifty-five samples per day. Great skill is acquired in thus estimating the value of the ore, the sampler being able to make from eight to ten valuations an hour.

As these are the only determinations made, it can be readily seen that the amount of the non-free-milling gold which enters the mill is not known. The percentage of sulphurets has been determined in several instances, and varies from 2½ and 3 per cent., to 6 and even 10 per cent. The assay-value of pure concentrates, freed from rusty gold or gold that is still enclosed in quartz (the ore being not sufficiently crushed), has lately been shown to vary from $4 to $90 per ton, the average for the district being about $25 per ton.

Sorting of the Ore.—As the gold is finely disseminated throughout

the entire vein-matter, comparatively little sorting in the mine can be expected. There occur, however, in many parts of the veins, igneous intrusions, locally called "porphyry," which form "barren horses." When the Nevada system of timbering in square sets was exclusively in use, no distinction was made between mill-rock and waste, but it was considered that the cheapest way to get rid of the latter was to run it through the mill. Lately, however, it has become the custom to fill the chambers formed by this timbering with waste rock and to hoist any excess of it to the dump, thus sending less waste through the mill; but large quantities are still got rid of in the old way, by running through the mill.

II. Enumeration of Mills.

When in 1876 the auriferous gravels and lodes were discovered, there was a "rush" to the Black Hills, and in a very short time a large number of mills sprang up. Of these the seven named in the following table are still running with 640 stamps.

Table I.—*List of Stamp-Mills.*

Name of Mill.	Year of Erection.	Location.	Name of Company.	No. of Stamps.
Homestake, .	1878	Lead City,	Homestake Mining Company,	80
Golden Star, .	1879	" "	" " "	120
Highland, . .	1880	" "	Highland " "	120
Deadwood, . .	1879	Terraville,	Deadwood-Terra Min'g "	80
Golden Terra,	1880	"	" " " "	80
Father de Smet,	1878	Central City,	Father de Smet " "	100
Caledonia, . .	1879	Terraville,	Caledonia " "	60*

The 25-stamp custom mill, the "Cassel" mill, which was built at Central City, about the same time as these, is running principally on cement gravel and will not be discussed here.

Although the mills given in the foregoing table are owned by five separate companies, they are all, except the last, under the management of the Homestake superintendent. Thus it follows that the working details of the six mills are all, as far as practicable, after one model—the Homestake. The Caledonia mill stands alone, and works on a very different rock. Therefore the details of its plant differ considerably from those of the others.

* To the Caledonia mill twenty additional stamps are being added. These will be running by the time this paper appears in print, making an aggregate, outside of the Cassel mill, of 660 stamps.

In the following pages the general arrangement of each mill will be given separately; but the details will be summarized under two heads, Homestake and Caledonia.

III. Outline of Plant and Process.

The crushing is done by means of rock breakers and stamps. The breakers reduce the coarse ore to a size suitable for the stamps. The ore, arriving at the highest level of the mill (the ore-floor) in mine-cars, is discharged from the side or the bottom of the car (dumping and bottom-discharge cars being both in use) over grizzlies to the crusher-floor; or it goes directly to the crusher-hopper. The small ore-particles, passing through the grates of the sizing-screen, and the coarse ore (which has been reduced in size by the crushers), both drop into the same ore-bin, which reaches down to the cam-floor. Here a number of chutes deliver it to the automatic feeders, each of which discharges its contents continuously into the mortar to which it belongs. Here the ore is pulverized by stamps (five in each mortar) lifted at regular intervals by corresponding cams, which are keyed to a cam-shaft, placed in front of the battery on the cam-floor. Water is fed continuously into the mortars and forms, with the ore, a liquid pulp, which passes through a screen at the front on to and over the apron-plates on the lower floor of the building. The Caledonia mill has blankets on the lower end of these plates to catch any coarse heavy particles; in the other mills the pulp passes directly from the apron-plates to the mercury-traps and through them on to sluice-plates. From the traps, placed at the end of these, the pulp runs into one main sluice which may again have one or more traps before the pulp is finally allowed to run to waste.

Thus the entire process of passing the auriferous coarse rock from the ore-floor to the final discharge, at the end of the main sluice, is an automatic one.

Battery-amalgamation is used to extract the gold. It begins in the mortar, where mercury is added at intervals (while the continuous fine crushing with the stamps is taking place) and ends on the apron-plates, where nearly all the amalgam not retained by the inside amalgamated copper plates is collected daily, any deficiency in the collecting mercury and amalgam on the plates being supplemented by the various traps.

As the mills on the "Belt" have to treat low-grade ores, it is necessary to their profitable operation that large amounts should be put through as rapidly as may be, and that, at the same time, as

much gold as possible should be saved by simple means. To effect this, a compromise is made between the two extreme methods of gold-milling. One of these aims at extracting as much gold as possible in the battery at the expense of capacity; the other, by amalgamating outside of the battery, increases the crushing capacity, but requires a number of expensive operations to recover the gold. In the Black Hills, amalgamation is carried on both inside and outside the battery, thus combining the simple way of recovering the gold from the first method with the large capacity of the second. The aim is to crush rapidly to the desired fineness and arrange the amalgamation so that it shall be adapted to the large amount of pulp produced.

IV. General Features of the Mills.

Table II. (p. 503) gives a comparative view of the dimensions, power, batteries, and product of the mills. The following is an explanation of the letters employed in this table:

A, Horizontal 155 H.P. engine, with Meyers cut-off. B, 300 H.P. Harris-Corliss engine. C, 350 H.P. Corliss engine, Fraser and Chalmers pattern. D, two 60 H.P. horizontal engines, with automatic cut-off; one driving forty stamps, the other now driving twenty, but soon to drive twenty more now erecting.

a. Length and width are always of mill proper, excluding engine-room.

b. The boilers in this table are all horizontal, tubular, 54 in. diameter, 16 ft. long, with 46 tubes of 3½ in. diameter. Steam pressure is kept at 90 lbs. per sq. in.

c. Made up as follows: Wrought-iron stem (length 14 ft., diameter 3⅛ in.), 340 lbs.; cast-iron head (height 18 in., diameter at top 9 in., at bottom 8 in.), 240 lbs.; cast-iron shoe (cylindrical but 8 in. high and 8½ in. diameter, tapering shank 4½ in. diameter at base, 3½ in. at top, 5 in. high—the shoe being made of white iron, chilled for 6½ in. from base, the next 1½ in. and the shank being cast in sand and cooled slowly), 140 lbs.; cast-iron gib-tappet (diameter at ends 9½ in., in middle cylindrical part 6 in., wearing faces 2½ in. thick, middle part 7 in. long, total length 12 in.), 130 lbs. Total, 850 lbs., stem, head, shoe, and tappet being proportioned as 34 : 24 : 14 : 13.

d. Batteries in two rows, back to back, leaving this space between the rows for ore-bins and feeders.

e. From June 1, 1887 to May 30, 1888, inclusive.

f. Estimated from report of Homestake Co., June, 1888, where product of 200 stamps in Homestake and Golden Star mills (under same management and similar conditions) is given as 243,355 tons for the year ending May 30, or 1,216.775 per stamp.

g. See Report of the Director of the Mint for 1887.

h. Will be increased to 20 in., as 18 in., which was large enough for 60 stamps, is not enough for 80.

i. Formerly 12, the mill having been originally built for 60 stamps.

k. Batteries in single line.

l. Batteries in two rows, but face to face, with this space between them.

m. Intentionally slower crushing on harder rock than the other mills.

n. For year ending April 30, 1888.

o. Excluding some concentrates from preceding year.

TABLE II.—*Dimensions, Power, Batteries, and Product.*

Mill.	Dimensions, *a*.			Boilers, *b*.		Engines.			5-Stamp Batteries.							Product.		
	Length, Feet.	Width, Feet.	Vertical dist. of Crusher—floor above bottom of ore bin. Ft.	Number.	Cords of wood per 24 hours.	Type.	Cylinder. Inches. Dia.	Cylinder. Inches. Stroke.	Number.	Distance between lines. Feet.	Weight of stamps. Lbs. *c*	Drop, Inches.	No. of drops per minute.	Tons crushed per stamp in 24 hours.	Tons milled in one year.	Bullion produced.	Average yield per ton.
Homestake,	100	80	14½	2	11	A	20	42	16	44½*d*	850	9	85	4.5	96,790*e*	$903,407	$3.71
Golden Star,	120	88¾	23½	4	14	B	20	60	24	36*d*	850	9	85	4.5	146,565*e*		
Highland,	112	92	22½	4	14	C	26	60	24	46*d*	850	9	85	4.5	146,013*f*	$410,953*g*	$2.82
Deadwood,	112	46	22½	2	11	A	20	42	16	*k*	850	9	85	4.5	216,361*f*	$639,229*g*	$2.02
Golden Terra,	112	46	22½	2	11	A	18½	42	16	*k*	850	9	85	4.5			
Father de Smet,	68	60	40	2	11	A	20	42	20	36*l*	850	9	85	4.5			
Caledonia,	84	46½		2		D	20	30	12	*k*	850	12	74	3.3	73,422*m*	$295,816*n*	$4.02*o*

Distribution of Power.—The methods employed in transmitting power from the engine to the different parts of the mill may be classed under three types, represented respectively by the Homestake, Golden Star, and Highland mills. The Caledonia may be placed with the Golden Star, and the Deadwood, Golden Terra, and Father de Smet with the Highland, notwithstanding local variations, especially such as are necessitated in the Father de Smet by the peculiar arrangement of its batteries.

1. In the Homestake mill, the continuation of the engine-shaft forms the one line-shaft of the mill and is placed on the battery sills. This is a cheap construction and gives a solid foundation for the boxes, in which the line-shaft rests. The shaft is kept in line by the even pull of the long belts placed on each side at an angle of about 30 degrees. The supposed disadvantage of being obliged to stop the entire mill, if anything happens to the line-shaft, is insignificant. When the mill was built in 1878, the line-shaft was not made of sufficient thickness, in consequence of which it became twisted and broke within two years. The new shaft, having the requisite strength, has been steadily in use ever since. For minor repairs, the position of the shaft is a disadvantage. It is in an obscure place where it escapes attention, particularly as everything has to be done by artificial light.

2. In the Golden Star mill, the power is transferred from a small main shaft to two line-shafts on the cam-floor, the main shaft being nearly on the same level. This arrangement is favored by builders on the Pacific coast. It is shown in the Providence mill, Nevada City, Cal., for description and drawing of which, see the paper of Mr. F. D. Browning on "Gold Chlorination in California," in the *School of Mines Quarterly* for May, 1884, also reprinted in the *Scientific American Supplement*, No. 445, July 12, 1884.

3. In the Highland mill (see Plate), the small main shaft is placed between cam-floor and battery-floor, and is connected with two line-shafts, placed on the battery sills, behind the mortars.

Both of these mills have two line-shafts, which is probably necessary considering the power that is required for a 120-stamp mill. The disadvantage is that the pull of the belt, on one side only of the shaft, has a tendency to draw it off the line, and also causes great wear and tear. As to placing the line-shafts in front of the batteries on the cam-floor, or behind them on the battery-floor, there is a diversity of opinion. The former arrangement, as at the Golden Star mill, gives the best light in the mill and makes the shafts easily

accessible. The power from the engine-shaft is transmitted to the line-shafts, and from these to the cam-shafts, by long belts, which are nearly horizontal. They require no tighteners, and last longer than those of either the Homestake or Highland mills. On the other hand, the boxes of the line-shafts rest on transverse sills on the cam-floor, which, although braced and strengthened in various ways, cannot furnish the same sure foundation as that at the Homestake and Highland mills. Experience has shown, however, that this disadvantage is more than counterbalanced by the smaller consumption of belts and the easy accessibility of the line-shafts. The Highland mill has the line-shafts on the sure foundation of the battery-sills; but the darkness and the exposure of the line-shafts, thus placed just behind the mortars, to the trickling of water and fine ore, are serious disadvantages. Again, the belts from the main shaft down to the line-shafts, and from these up to the cam-shafts, are short and steeply inclined, requiring powerful tighteners, which cause an increased consumption of belts. The mill, however, like the Homestake, is so arranged as to leave the battery, with the apron- and sluice-plates, free from any superstructure.

Particulars as to shafting, belts, etc., will be found in Table III. (p. 506).

The relation of the horse-power of the engine to each stamp averages for the seven mills about 1.7 : 1. This low figure is due to the large number of stamps in each mill (80 to 120), which is much higher than the common average in gold-mills.

Supply of Water and Fuel.—A regular supply of water is a prime necessity in milling. In this district it is chiefly furnished by two companies, at prices varying from 50 to 57 cents per stamp per day, the supply being brought in ditches. In winter, water becomes scarce, and then the mills of the Homestake management are supplied in part by pumping from the Homestake and Deadwood-Terra mines. This water otherwise runs to waste. The big Highland mill would be obliged to stop four months each winter if the tailings of the Homestake, Golden Star and Highland mills were not settled, and the clear water pumped back into the supply-tank. The method by which this is done is simple and effective. The tailings of the three mills are discharged together into the creek, called Gold Run. A little way further down, this broadens, and is closed by two dams, one below the other, forming two reservoirs. The upper overflows into the lower, which is four times its size, and this, in turn, has its overflow into the bed of the creek. The dams consist of cribbing,

TABLE III.—*Shafting, Breakers, and Belts.*

Mill.	Line-Shafts.			Rock-Breakers.		Character and Size of Belts.											
		Diameter in Inches.				Main-to Line-Shafts.			Line-to Cam-Shafts.			Line-to Crusher Counter-Shafts.			Crusher Counter-to Crusher-Shafts.		
	Number.	Near Engine.	At Opp. End.	Number.	Type.	Type.	Ply.	Width.	Type.	Ply.	Width.	Type.	Ply.	Width.	Type.	Ply.	Width.
Homestake.....	1	7	4½	5	E	.....	.....	.....	G	5	14	H	.....	18	G	4	9
Golden Star.....	2	9	4½	6	E	G	6	28	G	5	16	G	5	12	G	5	10
Highland.....	2	7	4	6	E	H	.....	41	G	5	12	G	5	12	G	4	12, 14
Deadwood.....	1	7	4	4	E	H	.....	41	G	5	12	G	5	12	G	5	12
Golden Terra....	1	7	4	4	E	H	.....	41	G	5	12	G	5	12	G	5	12
Father de Smet..	2	5½	3⅝	5	E	G	6	30	G	6	24	G	5	12	G	5	12
Caledonia.....	2	5 5/16	4½	1	F	G	5	28	G	5	14	G	5	12	G	6, 5	20–12

E = No. 5 Blake Rock-breaker. F = No. 6 Gates Rock-crusher. G = Rubber. H = Leather.

filled up solidly with waste rock. On their upper sides they are lined with water-tight planking. Down the middle of this, runs a wooden box, three sides of which are made of solid plank, the fourth being left open. When the reservoir is to be filled, this box is closed by pieces of heavy plank placed transversely. As far as the mere filling of the reservoir is concerned, the box might as well be closed by one solid piece, but the object of a number of pieces is to discharge the water gradually. This is done by removing the pieces, one after another, as the water is lowered, that the sands may be kept in suspension and carried through the culvert. Were the box opened at the bottom or to its full height at once, the sands would be carried into the culvert in such a quantity as to clog it. This culvert, in which the box ends, passes through the dam and under the lower reservoir and lower dam to the bed of the creek below. The lower dam is arranged in the same way. When the reservoirs are not in use, the water of the creek passes off through the culvert. When they are to be filled, the boxes are closed. The water accumulates in the upper reservoir until, after six hours, it overflows, leaving all the coarse sands in the upper reservoir, and carrying with it only the finer slimes, which settle in the lower one. From this the clarified water is pumped, at the rate of 60 cubic ft. per minute, into the Highland tank, 200 ft. above. The coarse sands are removed from the upper reservoir every twenty-four hours. In order to do this, the transverse planks closing the discharge are removed, one after another, and the water passes off, carrying the sands with it. As this process takes four hours, and the filling six, there are fourteen hours of overflow into the lower reservoir, where the slimes settle. These are removed once in two months, in the same way.

The fuel for the mills under the Homestake management is supplied by the "Black Hills and Forte Pierre Railroad Company." This road, with about thirty miles of 3-ft. gauge track, runs along the divide between Gold Run and City creek, terminating at a point about fourteen miles south of Lead City. The whole section was originally heavily wooded, but has been quite denuded by the constant demands made upon it. The railroad is very winding and is quite a feat of engineering. It runs down the slope into Whitewood creek, and up the opposite height, until it finally reaches the point where timber is still to be obtained. This road, as soon as spring opens, is employed in transporting the wood and timber which have been cut and stored along its line, and is in constant use until the heavy snows block it for the winter, generally from January till

April. It has three branches, towards the three towns where the mills are situated, and communicates directly with these by means of long wooden chutes, down which the wood is discharged. These are from 700 to 1500 ft. long, running down the slope of the mountain. The chutes, 25 in. broad by 12 in. deep, are made of 4-in. plank. The bottom and 9 inches of the sides are lined with ¼-in. iron. The fall of the chute is 6 in. to the foot until the curve begins, when it is 4½ in. This continues to the nozzle which is elliptical. When the chute is in use, a small current of water is passed through it to prevent the iron from becoming too hot, and also to act as a lubricator. The cord wood, unloaded into the top of the chute, passes down the incline with great velocity. At the nozzle it is deflected from its course, and, through the momentum obtained in its downward passage, shoots up into the air and drops some distance off on the wood pile. In order to discharge the wood on a somewhat large area, the nozzle is made movable. The Caledonia mill has its wood hauled by teams, but is making experiments with soft coal, as the price of a cord of wood in the district is six dollars.

Auxiliary Arrangements.—As the thermometer often sinks to forty and fifty degrees below zero (Fahrenheit), the mills and the water used in them have to be warmed. This is generally done by the use of steam. Along the ground-floor, in front of the apron-plates, there runs a sheet-iron drum, 18 in. in diameter, through which the waste steam of the engine is conducted. The battery-water is prevented from freezing by steam coils passing through the supply-tanks.

Fire-plugs, with the necessary hose-attachments, are placed at regular intervals, to be used in case of accident.

To reduce the cost of repairs and renewals, which is so large a part of the expenses of gold-milling, the Homestake company has a foundry, where the rock-breaker shoes and dies, pitmans and toggle-plates, mortars and dies, boss-heads, tappets, thimbles for props, cams and hubs of cam-shaft pulleys, and all shaft-boxes, are cast, from No. 1 and No. 3 foundry-iron and worn-out castings. These are also sometimes bought at one cent a pound. The casting is all done in sand, with the exception of the rock-breaker shoes and dies and the faces of the battery-dies, which are chilled.

In the machine shop, which is a very complete establishment, all the necessary repairs are made. This is the only good shop in the district, and does the necessary work for outside companies also.

V. The Mills in Detail.

The six mills under the Homestake management (that is, all the mills here described, except the Caledonia) have the same patterns for all parts needing frequent renewal. This greatly reduces the amount of material kept on hand and the labor and cost of repairs.

Grizzlies.—These are coarse screens, upon which the ore, arriving from the mine, is dumped and separated, the screened fines dropping directly into the bins. This relieves the breaker of ore that does not need to be crushed, and only the coarse ore has to be shovelled into it. They are 3 to 4½ ft. wide, 10 to 14 ft. long, and set at an angle of about 40 degrees, representing a rise of 7 to 8 in. per ft. They are made generally of wrought iron bars, 1 in. wide and 2 to 4 in. deep, held in place, 1½ to 2 in. apart, by three or four, sometimes five, 1-in. iron rods, provided with washers at the proper intervals. Mr. Bowie* gives 2040 pounds, as the weight of a grizzly at the Father de Smet mill, 4½ x 12 ft. in size, with 24 bars, 1 in. wide by 2 in. deep, and 1½ in. apart. The grates last about four years. In some instances old wrought-iron rails, with base turned up, take the place of the standard rectangular bars; but they do not last much over one year.

Rock-Breakers.—All the Homestake mills use the Blake. The coarse ore, rolling down from the grizzlies, collects on the crusher-floor, and is fed by hand into the mouth of the breaker on the same level. From the crusher, it passes into the bins which have already received the fine ore from the grizzlies. The No. 5 breaker used (the largest pattern in the market) has a receiving capacity of 9 by 15 in., is set to crush from 1½ to 1¾ in., and is run 20 hours out of the 24, each crusher being calculated to serve 20 stamps. The amount crushed per day is not known, as there is no easy way of determining how much of the ore in the bins has passed through the crusher. If one-fourth of the ore (a probable proportion in view of the slaty character of the ore) passes through the screens, escaping the breaker, and if 20 stamps crush 90 tons of Homestake ore in 24 hours, the amount crushed in 20 hours by one crusher is 67½ tons, or 3.4 tons per hour. This small figure, as compared with the capacity of the crusher, which is given at 7 tons, is due to the smallness of the mouth of the crusher, which necessitates the breaking-up of the ore—a serious matter, occasioning much delay. No part of the mill-work is so laborious as this breaking and feeding of the ore, which has to be done by hand.

* *Trans.*, x. 98.

The wear and tear of material in a crusher is comparatively small, a shoe lasting two months, a die four months.

The small receiving capacity of the No. 5 Blake is a marked disadvantage to it, in comparison with the Gates crusher, lately introduced at the Caledonia. The ore of the Caledonia mine breaks rather coarse (that is, it does not show much of the slaty character of the Homestake ore), and is dumped immediately into the hopper of the crusher without use of the grizzly. With about the same horse-power as three No. 5 Blakes, and set to the same size, one No. 6 Gates crusher (with three receiving openings, each 12 by 18 in.), attended by one man only, crushes 200 tons in 10 hours. When the three Blakes were in use, it required 20 hours and 5 men to produce the same result. Mr. T. L. Skinner, the superintendent, says in his last report that this new crusher saves him $27 a day. In order to make the iron head last longer, he uses three sets of concaves of graduated thickness, the thinnest first, and so on. When, after some time, the head and first set of concaves have become so worn away as to increase the width of the discharge above 1½ in., the second set is introduced; and, when they in turn wear thin, the third. By the time these are worn down (after about five months), it becomes necessary to replace the head. The disadvantage of the Gates crusher is its enormous weight (No. 6 weighs 30,000 lbs.), and the consequent difficulty of transporting it, especially in some mining regions.

The best arrangement for a large mill seems to be to use a still larger Gates crusher (No. 8, with receiving openings 18 by 48 in.), set to crush coarse, and discharging into two No. 6 crushers, set to crush fine. Thus the largest pieces of rock any man could handle would pass directly into the crusher, and the breaking by hand in mine and mill would be reduced to a minimum.

The smaller Gates crushers, with correspondingly small mouths, are not to be preferred to the Blake crushers.

Ore-Bins.—These receive the ore from the grizzlies and crushers, directly over them, and discharge it through chutes into the hoppers of the feeders. They are triangular, with one vertical side, facing the battery and reaching down to the cam-floor. Just above the latter are the openings (one for each feeder) through which the ore passes downward into the chutes, terminating in the hoppers of the feeders. The quantity of the discharge is regulated by a sliding door. In a double mill, the inclined bottoms of the two bins diverge, leaving an open space between them which has the shape of an in-

verted V. This arrangement of bins is common to all the mills except the Father de Smet.

The bottoms of the bins, 3 in. thick, are made of 1-in. board, running lengthwise, with 2-in. plank, placed at right angles upon it crosswise. The bottom and sides are carefully braced with strong beams. There are no separate compartments or special arrangements for directing the ore towards the discharge-openings. The descending ore soon wears out such contrivances. It is advisable to line with iron the upper part of the bottom, on which the ore drops from the grizzlies and crushers. Otherwise, it wears out much faster than the middle and lower parts, which last 5 to 6 years.

It is best to make ore-bins as large as practicable, so that, in case of accident in mine or at rock-breakers, the mill need not stop. The capacity ought never to be less than one full day's supply. By multiplying in each of the three double mills, the horizontal distance between the two sets of batteries into the vertical distance between crusher-floor and cam-floor, the comparative size of their ore-bins can be approximately estimated. The following table shows the result. It is assumed that the distance between front of bin and battery, as well as the incline of the bin, is about the same in all.

TABLE IV.—*Comparative Capacity of Ore-Bins.*

Name of Mill.	Horizontal distance, feet.	Height, feet.	Product, square feet.
Homestake, ...	44.5	14.25	634
Golden Star, ...	36.0	23.75	855
Highland, ...	46.0	22.75	1046

This would show that, of these three double mills in which the batteries are arranged back to back, the Highland has the largest bin-capacity. In the Father de Smet, where the batteries discharge towards the center, the bins, built entirely above the batteries, and extending to the side-walls of the building, have a still larger capacity, the figures, corresponding to the dimensions given above, being 57 feet by 30 feet, or 1710 square feet. There are, however, decisive objections on other grounds to this arrangement. The apron-plates are so overshadowed by the inclined bottoms of the ore-bins above that the facility of supervision, claimed as one advantage of this plan, is largely neutralized by the prevailing darkness, even at noonday.

Feeders.—The working capacity of a battery, and its exemption from unnecessary wear and tear, depend greatly on regular and equal feeding of ore. This used to be done by hand, but is now generally accomplished by automatic feeders, placed at the back of the batteries, and discharging either directly into the feed-opening of the mortar (the lip of the feeder reaching into the mortar-feed), or, as at the Caledonia, upon a small inclined iron-lined apron which leads to the mortar. By the latter arrangement, a little more room is left between feeder and mortar, and the feed-opening can be narrower and longer, and the ore more uniformly distributed under the stamps. Thus, the Homestake mortar feed-opening is 24 in. long and 4½ in. wide, while that of the Caledonia, occupying the entire length of the mortar, is 52 in. long and only 3 in. broad.

The two ore-feeders, used in the district, are the "Hendy Challenge" and the "Tullock Automatic." Each has a well-earned good name, the Challenge being, perhaps, more desirable for wet ores. Both right-hand and left-hand feeders are used, the bumper-rod standing between stamps 1 and 2 or 4 and 5. The Challenge feeders, introduced with the new 20 stamps of the Caledonia mill, are central feeders, the bumper-rod being placed next to the central stamp. The rod is guided from the cam-floor, to which a piece of board, with suitable hole for the rod to pass through, is fastened.

The comparative advantages of the two-feeders may be summed up by saying that, while the sheet-iron plate below the hopper of the Tulloch wears out quickly (with Homestake ore, in two years), it is cheap and can be patched or renewed by any blacksmith; while the circular cast-iron carrier-table of the Challenge lasts seven years with the same ore, but is costly, and, if anything is out of order with the gearing, it requires a shop and a machinist.

Battery-Foundations, Frames and Guides.—The essential importance of a good foundation is well known. In preparing it, a rectangular pit, from 11 to 14 ft. deep, is first dug to receive the mortar-block. It is made sufficiently long and wide (4 by 6½ ft.) to leave a space of about 24 inches all around the block. The bottom is then carefully levelled and some sand tamped down. On this are placed two layers of 2-inch plank, spiked crosswise to each other, and then the planks which form the mortar-block. The latter used to be placed directly on the bottom of the pit, the uneven tops being sawed off afterward. Now, care is taken that this 4-inch wooden floor shall be accurately horizontal, and that the distance between it and the bottom of the mortar shall have the length of

the mortar-block. The top of the block is only planed off, which saves time. By the use of this flooring, the time required for replacing a mortar-block is reduced from six or seven days to five. The mortar-block consists of planks, from 11 to 14 ft. long (according to the depth of the pit), of varying breadth, and not more than 2 or 3 in. thick, as it is difficult to find wood of greater thickness which is sound throughout. They are spiked together, and are fastened above and below with binders, bolted to each other by transverse rods, the upper binders (8 by 12 in.) being even with the top of the mortar-block, and the lower binders (12 by 12 in.) 3 ft. lower down. The space around the mortar-block is then carefully filled and tamped with rock and tailings up to the level of the mud-sills, which are about 4 ft. below the bottom of the mortar. When the top of the mortar-block has been planed off and levelled, a sheet of rubber cloth, $\frac{1}{4}$ in. thick, is placed over it, and the mortar put in place. Through the four holes in the flanges on each side, pass eight bolts, from 3 ft. to 4 ft. 6 in. long, and from $1\frac{3}{8}$ to $1\frac{1}{2}$ in. in diameter, with which the mortar is fastened to the block.

In placing the planks forming the mortar-block, and in adjusting the bolts that hold down the mortar, a decided improvement has lately been made at the Homestake mills. The planks which always stand upon end, were formerly so spiked that their width was parallel to the short side of the mortar. The holes for the eight bolts were then bored into the mortar-block from above; at a suitable distance below, recesses were chipped out to receive the nuts which secure the lower ends of the bolts. Now the planks are so placed that their width is parallel to the long side of the mortar. The bolts have only at their upper end a thread, and end in a loop at the bottom. Through these loops and through the mortar-block, pass horizontally 2-in. iron rods. The planks, on the two sides of the block, where the bolts pass down, are cut out to receive them. The advantages of this arrangement are apparent. In addition to the mortar being more securely and evenly tied to the block, it is easier to renew the mortar-block if necessary. The pit need only be dug in the front of the mortar, and when the front binders have been removed, it is easy to tear out the planks, one after another, with pick and adze. In putting in the new block, the two outside rows of planks, with places cut to receive the bolts, are kept ready, so that only four horizontal $2\frac{1}{2}$-in. holes for the rods need be bored when the planks have been spiked together. With the old method, two trenches had to be dug, instead of one, if a mortar-block was to be

exchanged. Then the planks had to be chipped out in pieces until room enough had been made for them to be torn off. This cost much labor and time.

Battery-frames.—These are generally 12 by 24 in. in size, with recesses for the boxes of the cam-shaft. They are placed on the short sides of the mortar, and are independent of the mortar-block, standing on the cross-sills (which are placed on top of, and at right angles to, the mud-sills), and tied by the upper and lower guide-timbers, and, at the foot, by two longitudinal beams, bolted to them, and let into the cross-sills. The frames are braced either from the back or the front of the battery, according to the way in which the power is transmitted to the cam-shaft. In the Homestake, Highland, Golden Terra and Father de Smet mills, where the line-shafts are at the back of the battery, the frames are braced by posts, generally placed on an incline between frame and cross-sill, leaving the entire front of the battery unobstructed. In the Golden Star and Caledonia mills, the cam-shafts receive their motion from line-shafts placed on the cam-floor, and the frames are braced against the cross-beams to which the boxes of the line-shafts are bolted.

Guides.—The stamps are held upright by two sets of guides, fastened to the guide-timbers which tie the battery-frames. The upper guides are above the tappets, the lower ones between the cam-shaft and the top of the mortar. At the Homestake mill, the center of the lower guides is 17½ in. above the top of the mortar, and the cam-shaft, revolving between the two sets of guides, is 3 ft. 10½ in. below the center of the upper, and 4 ft. above the center of the lower guides, making the total distance between the two sets 7 ft. 10½ in. At the Caledonia mill, the distance from top of mortar to center of lower guides is 16½ in.; from these to center of cam-shaft, 3 ft. 4½ in.; from this to center of upper guides, 4 ft. 2½ in.; total, 7 ft. 6¾ in.

Each set of guides consists of two pieces of 4-in. pine plank, 16 in. deep, having semi-circular grooves, through which the stems of the stamps pass. Between the two parts of the guides, when new, are inserted small pieces of wood to hold them slightly apart. When the grooves are somewhat worn, these slats are removed, and the guides brought nearer each other. Later on, the faces are planed off to diminish the size of the grooves, so that the stamps may not be held too loosely. Each set of guides is secured to the guide-timber with eight ⅞-in. bolts. The grooves in the guides are lubricated with a preparation of black lead and linseed oil, mixed warm

in such proportions as to form a soft paste. Oak guides last eighteen months; pine, only four months.

Mortars.—Two kinds of single discharge mortars are used in the district. Each consists of one solid casting. The bottom and sides are so thick as not to need any lining, this thickness decreasing very considerably as the feed is approached. The outside walls are vertical, with the exception of the discharge, which projects somewhat. The top is closed by two pieces of 2-in. plank, which rest on lugs, $\frac{3}{4}$ in. wide, cast in the mortar, 2 in. below the top. These planks have each five semi-circular recesses which, when placed together, form holes for the passage of the stems. In addition to these five large holes, two smaller ones are bored for the two 1-in. water-supply pipes, placed between stamps 1 and 2 and 4 and 5. Two mortars are placed close together, as the stamps of both are set in motion by one cam-shaft. In order to reach the mortars, etc., a passage-way is left between every two pairs of batteries. The water-supply is derived from a 3-in. main, running along the front of the batteries. From it passes upward a 2-in. pipe between each pair of batteries. With this is connected a 2-in. horizontal pipe, from which four 1-in. pipes branch off at right angles, two for each mortar. In addition to this water-supply, there is a 1-in. pipe at each passage-way, close to the mortar, coming from the 3-in. main. A hose is attached to clean the apron-plates and for other purposes.

The points of difference between the two mortars lie in the inside dimensions of the lower part of the mortar and in the arrangement and number of the inside amalgamated copper plates.

1. The Homestake mortar (Figs. 1, 2 and 3), weighing 5400 pounds, is $54\frac{1}{2}$ in. high and $54\frac{1}{4}$ in. long. The feed-opening, beginning $6\frac{1}{2}$ in. below the top, is 24 in. long, $4\frac{1}{2}$ in. wide and 7 in. deep. On entering the mortar it remains 24 in. long and 7 in. deep. At the bottom of the feed, forming the continuation of the incline over which the ore passes into the mortar, is a lip, $4\frac{1}{4}$ in. wide and $1\frac{1}{4}$ in. thick, designed to discharge the ore against the upper half of the stamp-head. The lower edge of the lip is 14 in. above the inside bottom of the mortar. As the lip wears out fast, it might be well to cast it thicker, as has been done in the Caledonia mortar. Taking the front view of the mortar, we find, $15\frac{1}{2}$ in. from the bottom, the discharge-opening, $48\frac{1}{2}$ in. long and $21\frac{1}{4}$ in. high. The frame is inclined outward about 10 degrees from the vertical. On the short sides of the discharge-opening, are grooves to receive the chuck-block, screen-frame and curtain, which are held in place by

Fig. 1.

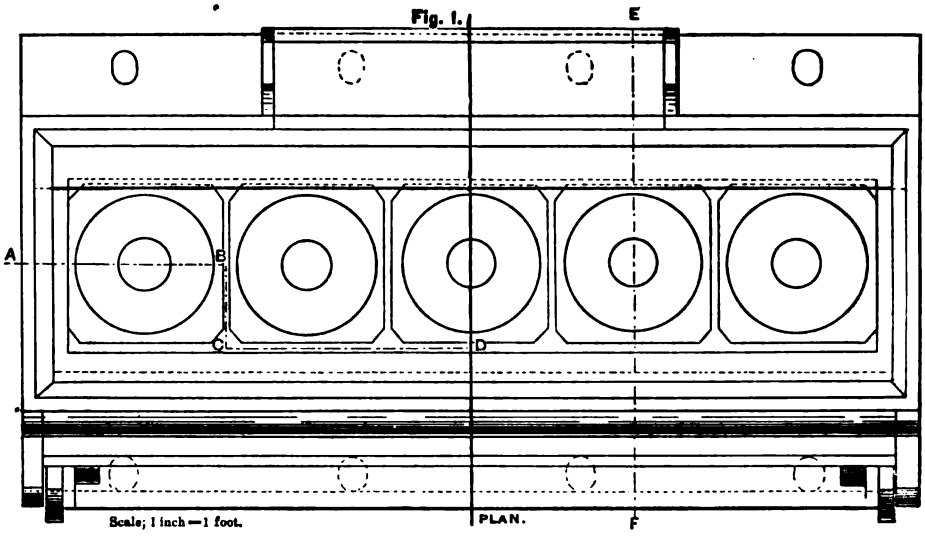
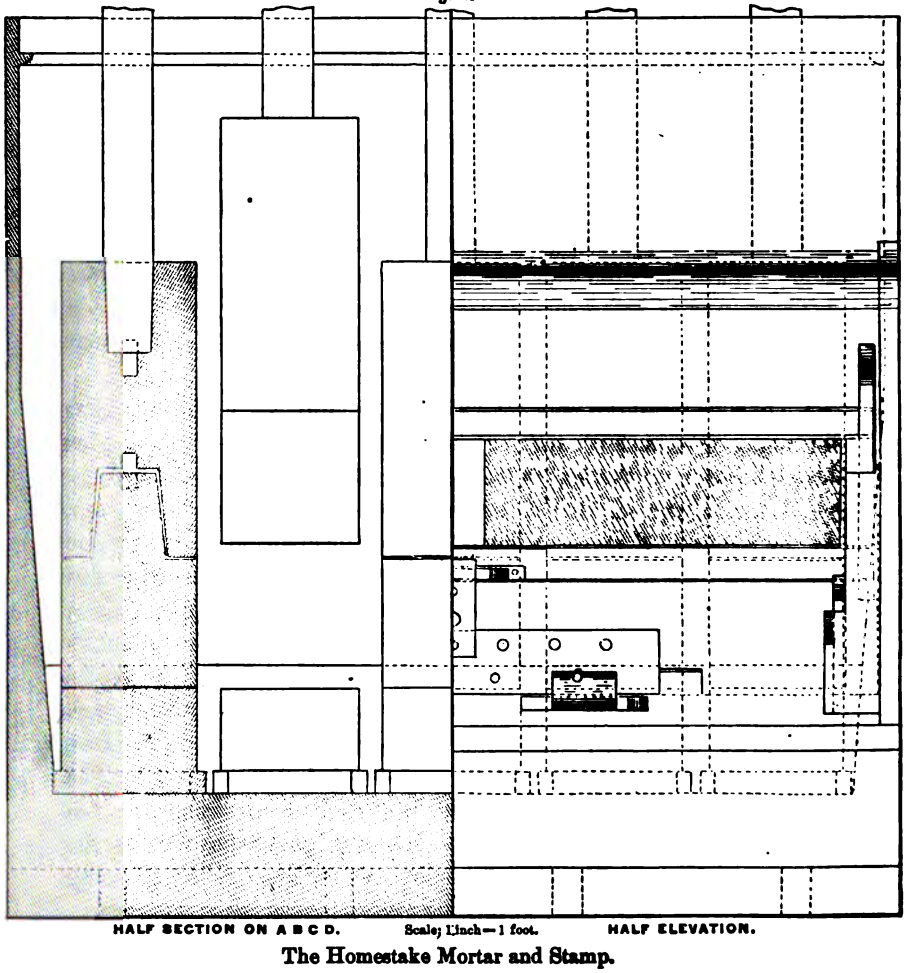
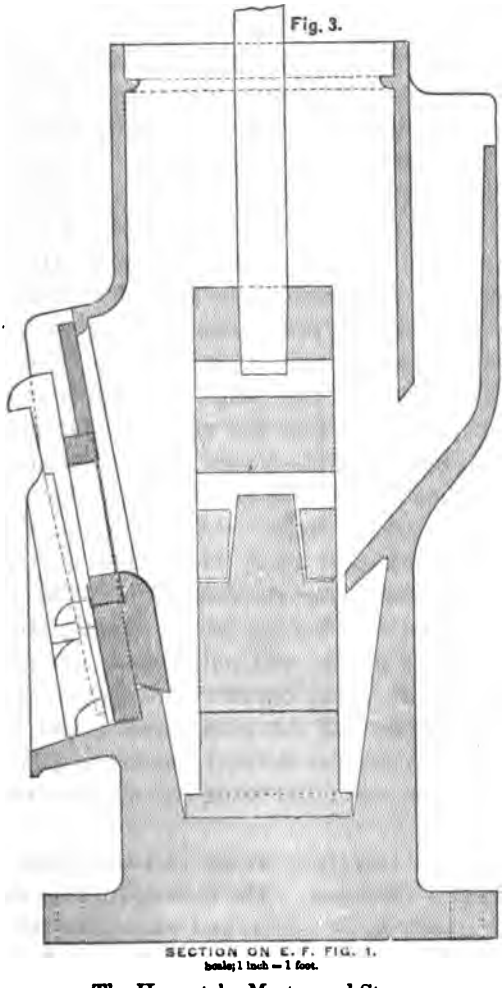


Fig. 2.



The Homestake Mortar and Stamp.

keys and sockets. The chuck-block is also fastened at the bottom by two horizontal keys, supported by lugs on the outside lip of the mortar below the discharge. Viewing the mortar in cross-section, we first have the two bottom flanges, 3 in. high and 5 in. broad.



The Homestake Mortar and Stamp.

The bottom of the mortar (the mortar-bed) is 7½ in. thick, the sides, at the foot of the dies, 3½ in. The inside dimensions are: Width at the bottom, 10½ in.;* length, 50 in.; height to issue of mortar

* Formerly the width was 14 inches, but, in order to increase the crushing capacity, it was reduced.

(not of pulp), 8¾ in.; width at this point, 13½ in.; at the top of discharge-opening, 20 in.; at the top of mortar, 16 in.; total inside height, 47 in. The casting is ¾ in. thick, from the top down to the feed-opening, on three sides, the back being a little thicker.

A mortar lasts four years, wearing pretty uniformly at the sides and back.

2. The Caledonia mortar weighs 5700 pounds, is 57½ in. high and 54 in. long. The feed-opening, beginning 15½ in. from the top, is 3 in. wide, 11 in. deep and extends the entire length of the mortar, having a strengthening rib in the center. At entering the mortar, it is 50½ in. long and 7½ in. deep. Here the lip, 2½ in. thick and 8 in. wide, measured on the incline, begins. The bottom of the lip is 15 in. from the foot of the dies. As in the Homestake mortar, the ore is discharged towards the head of the stamp. The lip serves also as a protector to the amalgamated copper plate below it. The discharge-opening in front, 50 in. long by 17 in. high, begins 20 in. above the bottom of the flange. Its frame is also inclined outward about 10 degrees from the vertical. The grooves on the sides, receiving only the screen-frames and the curtain, are simpler in construction than those of the Homestake mortar. The lugs for the horizontal keys are the same. Taking the cross-section, we find the flanges 3 in. thick and 4½ in. wide. The mortar-bed is 7 in. thick, the sides, at the foot of the dies, 4¼ in. The inside dimensions are: Width at the bottom, 10 in.; length, 50½ in.; height, 14 in. to the issue of mortar and pulp, where the width is 16 in. This increases to 19 in. at the top of the discharge. The top of the mortar is 13½ in. wide, and the total inside height 50½ in. The casting, from the top down to the feed-opening, is ¾ in. thick.

A mortar lasts six years, and wears out more on the short sides than at the back.

In comparing the two types, we see that they differ in the feed-opening, as already discussed. The feeding-lip also differs, that of the Caledonia mortar being thicker and wider than the other. The increase of width is necessitated by the presence of the amalgamated copper-plate below the lip; the mortar itself is also wider at the issue for the same reason. The depth of the Homestake mortar is 8¾ in. and that of the Caledonia mortar 14 in. The latter corresponds with the height at which the issue of the pulp occurs. In the Homestake mortar the issue is raised by the insertion of the chuck-block 16¼ in. above the foot of the dies, thus giving, with a shallower mortar, a deeper issue of pulp than the Caledonia mortar.

Dies.—The Homestake management casts its own dies. The quality of iron used is between gray and mottled, the top of the cylindrical part being chilled. The foot-plate has bevelled corners, and is 10 in. long, 10½ in. wide and 1¼ in. thick. The cylindrical part or "boss" is 9 in. in diameter and 5 in. high. The level of the die is 10 in. below the discharge which takes place over the chuck-block. The die weighs 121 pounds (one-seventh of the weight of the stamp) and lasts about six weeks, crushing 189 tons. By that time the cylindrical part has become slightly convex, and is worn down to 2 in. from the foot-plate. Its weight has then been reduced to about 30 pounds; thus 48 pounds of iron are consumed for every 100 tons of rock that are crushed.

The Caledonia mill buys its dies outside. They are of chilled white iron. The foot-plate has also bevelled corners, is 10 in. wide by 9½ in. long and 1½ in. thick. The cylindrical part is 8 in. in diameter and 5½ in. high. While the dies in the Homestake mortar fill its bottom completely, those of the Caledonia fit perfectly in the width only, there being a 3-in. space in the length that has to be divided up between the five dies. The distance from bottom of screen to top of die is 6 in. The die weighs 160 pounds (about one-fifth of the weight of the stamp) and lasts three months, crushing 300 tons of hard rock. The cylindrical part is then worn down within 1 in. of the foot-plate. The worn-out die weighs 38 pounds, making the consumption of iron 40 pounds for every 100 tons of rock.

Amalgamated Copper Plates are placed along the entire length of the mortar. In the Homestake mortar, one plate is set to the discharge opening; in the Caledonia mortar, there are two plates: one under the discharge, the other beneath the lip of the feed-opening.

The Homestake mills use the so-called chuck-block (half elevation, Fig. 2 and Fig. 3), placed against the lower flange and the two side-flanges of the discharge. The chuck-block consists of a 2-in. plank, bolted to the back of a 1¾-in. board, and extending from 2 to 2½ in. above it. Its inside upper edge is rounded off, and over this, and along the inside face, a $\frac{3}{16}$-in. copper plate is fastened with iron screws. The recess formed on top of the front board, 1¼ in. wide and from 2 to 2½ in. deep, is taken up by the lower part of the screen-frame. Between this and the front board is placed a strip of carpet to form a tight joint. The frame is held in place by a vertical piece of flat iron bolted to the center of the front board, a horizontal wedge being driven between the two. The front

board has an iron facing along its lower half, and two vertical strips towards the ends, to protect the wood against the two horizontal and the two vertical wedges with which it is fastened to the mortar. To the back (beneath the 2-in. plank having the sheet-copper), is tacked a strip of rubber cloth, which helps to make a tight joint between wood and flange of mortar. Two chuck-blocks, of different heights, are in use. When the dies are new, a chuck-block 7 in. high is inserted; when they have worn down 2 in., another chuck-block, 5 in. high, replaces it. Thus the height of discharge is kept nearly uniform. The distance between the face of the shoe and the chuck-block (2 in.) is rather small. The violent motion of the battery-water drives the sands against the copper plate and scours off amalgam that has been caught on it. Thus comparatively little amalgam can settle on the plate. Wooden chuck-blocks last six months. After this time the coppers have to be removed and put upon new blocks, or they are scraped very carefully, put aside, melted and sold. Mr. R. Graham, the millwright of the Homestake mill, has therefore replaced, in his mill, the 2-in. plank, to which the copper plate is screwed, by $\frac{1}{4}$-in. iron, to which the $\frac{3}{16}$-in. copper plate is riveted with copper rivets. The face of the $1\frac{3}{4}$-in. wooden front board is covered with $\frac{1}{8}$-in. iron; thus the distance between face of shoe and Graham chuck-block is $3\frac{5}{8}$ in. instead of 2 in. This iron chuck-block lasts as long as the mortar, and more amalgam collects on it than on the wooden one. Of the free gold recovered, 55 per cent. is caught on this inside plate.

The reason that the Caledonia mill has amalgamated copper plates at both front and back, is that the ore milled is not oxidized at all, which makes it harder to extract the gold. The aim is to keep the pulp longer in the battery and thus counteract the refractory character of the ore. The plate in front is 5 in. broad, the one at the back, 8 in. Both are made of $\frac{3}{16}$-in. copper and are simply bolted to the mortar, the lower edge of the plates being 9 inches above the foot of the dies. Of the free gold recovered, 60 per cent. is caught on these inside plates.

Screens.—Both diagonal-slot and wire screens are used in the district. With the exception of the Father de Smet mill, which uses partly No. 30 brass-wire screens, all Homestake mills use diagonal-slot screens made of heavy Russia iron. The needle number is 7, corresponding to a 30-mesh wire screen, the width of the slot being 0.024 in. The thickness of the iron is No. $24\frac{1}{2}$ (American wire-gauge) and its weight 0.987 lbs. per sq. ft. The slots are $\frac{1}{2}$ in.

long and there are eight (formerly only seven) to the inch. The punched surface of the screen is 48 x 7 in.; there is a margin of 1 in., making the entire screen 50 x 9 in. A screen lasts two weeks. The wooden frame is 4 ft. 4 in. long and 11½ in. deep, and has a strengthening rib, 6½ in. long, down the center. In fastening the screen to the frame, the lap is first tacked on, to hold it in place, then a piece of rubber cloth, 2 in. wide, is placed over it, small holes are punched through rubber and lap of screen and both are nailed to the wooden frame. The screen is placed on the frame with the rough side facing the mortar. On the outside of the frame are fastened, by means of two screws, three iron facings, ½ in. x 9 in. and $\frac{3}{16}$ in. thick, which protect the wood from the one horizontal and the two vertical keys that serve to wedge the frame against the chuck-block and the planed flanges of the discharge.

Some time ago, experiments were made with screens of aluminium-bronze, which proved extremely satisfactory. The bronze contains, according to a letter from the Cowles Electric Smelting and Aluminium Co., 5 per cent. aluminium, 95 per cent. copper and a trace of silicon, and is furnished in unperforated sheets at 45 cents per pound. When new, it has a golden color, which it loses with use. The width and length of the slots are the same as in the ordinary screen, but there are nine slots to the inch instead of eight. The sheet is 0.035 in. thick. This screen lasts six months and does not break (while the Russia-iron screen breaks in two weeks). The wear is uniform over the entire surface, the slots enlarging to No. 5½ needle. The screens are then past use. The bronze, however, is not lost, but can be melted down and made into new screens. It is the intention of the Homestake management to introduce this screen throughout all its mills. This would have been done already, had not the contracts for Russia-iron screens been made before the bronze screen was tried.

The Caledonia mill uses No. 24 brass-wire screens, the thickness of the wire being No. 26, and the screening surface 48 x 5⅜ in. The screen lasts one week. It is fastened to a simple wooden frame, 53 x 12½ in., the horizontal sides being 3½ in. wide, the vertical sides 2½ in. Three wooden ribs, 1 in. wide, divide the screen-surface into four panels and thus prevent it from bulging out. The fastening of the screen to the frame and the wedging of this against the mortar are done as at the Homestake, except that there the screen-frame is placed on the chuck-block, and here it is keyed against the lower rim of the mortar discharge. The Caledonia uses

wire-screens because, although its stamps drop 3 in. farther than those of the Homestake mills, the splash is not so great, by reason of the greater width of the mortar and the space taken up by the amalgamated copper plate below the feeding-lip. The force of the splash in the narrow Homestake mortar is thrown entirely against the screen; that of the wider Caledonia mortar is divided between the screen in front and the recess at the back. Thus the slot-screen would clog, while the wire screen allows the pulp to pass through free.

The upper part of the discharge of both classes of mortars, above the screen-frames, is closed either by an inch board or by a canvas curtain, or piece of old belting, suspended from a lath. This curtain or belt hangs down and meets the screen in the mortar. It has the advantage over the board that the amalgamator can readily pass his hand into the mortar and remove any chips of wood, etc., floating on the water or adhering to the inside of the screen.

We have seen that in the Homestake mortar the pulp is driven with some force against the screens. In order to break its fall upon the apron-plate, a splash-board is fastened to the frame of the latter, thus preventing any amalgam collected there from being washed away. The Caledonia mortar has no splash-board, as the pulp does not pass the screens with sufficient force to endanger the amalgam collected at the head of the apron-plate.

Stamps.—The stamps (Figs. 1, 2 and 3) used by both companies are of the ordinary pattern. They weigh 850 lbs., about 16 lbs. to the sq. in. of crushing-surface, and their centers are 9¼ to 10 in. apart. For dimensions and other particulars as to the different parts, see the table already given.

The stem tapers 6 in. at both ends, so that when it breaks, usually where it enters the head, it may be reversed. At the Homestake mills, a stem lasts about three years before new ends have to be welded to it. The Caledonia mill requires for its 60-stamp mill five stems yearly.

The tough cast-iron head is without the wrought-iron rings which are often used at the upper or lower end. It has the usual key-ways for the removal of stem and shoe. These are parallel instead of being at right-angles, as is usual. To fasten the head to the stem, the latter is let down through the guide-holes and the socket of the head placed directly beneath it. The stem is lifted and dropped and driven with a hammer, if necessary. Then the stem and head together are dropped several times on a piece of timber placed be-

neath, until they are quite firm. At the Homestake mills, a head lasts five years, after which the socket has become so enlarged that the stem cannot be securely fastened in it any longer. The Caledonia mill requires one head a month.

Both managements buy their shoes from outside foundries. To fasten the shoe to the head, the shank is surrounded by small wooden wedges tied around it with a string; the shoe is put in place; and stem and head are allowed to drop. Thus the shank is wedged into the recess of the head, and by raising the whole and allowing it to drop several times, the shoe becomes thoroughly fastened in. To prevent it from injuring the die, a piece of planking is placed on the latter. At the Golden Star mill, a very neat arrangement is in use for fastening the wooden wedges to the shank of the shoe. After the wedges have been placed around the shank, a strip of canvas is wound around them and tacked to each wedge, forming a sort of collar, which can be easily slipped over the shank of a new shoe. This simple arrangement saves a great deal of time and labor on clean-up days, when a good many shoes have to be replaced.

After being some time in use in the battery, a shoe becomes slightly concave at the base, but wears, on the whole, more evenly than the die. At the Homestake mills, a shoe lasts two months, crushing 270 tons of rock. It is then worn down to 2 inches from the base of the shank and weighs 40 pounds. This corresponds to a consumption of 37 pounds of iron for every 100 tons of rock crushed. At the Caledonia mill, a shoe lasts three months and crushes 300 tons. It is replaced when worn down to 1 inch and weighs 35 pounds, which corresponds to a consumption of 35 pounds of iron for every 100 tons of rock. It may be asked why the Caledonia mill, which buys both its shoes and dies, does not use steel, which is so much more durable, and consequently cheaper, especially if the heavy freight charges are taken into consideration. The answer is that because steel lasts so much longer than iron, steel shoes and dies would continue in use after they had become uneven. This would reduce the crushing capacity of the battery, and thus in the end would be no saving at all.

In order to secure the tappet to the stem, the wrought-iron gib is first put into the recess. The tappet is then slipped over the stem, and when it has reached the desired place, is secured by keys which pass at right angles between the gib and the iron shell of the tappet surrounding the stem. All the tappets of the Homestake mills have two keys; those at the Caledonia have some two and some three.

Both faces of the tappet are used as working faces. In wearing down they become uneven and ridges are formed. The tappet then has to be removed and planed off on a lathe before it is fit for further use. Once in three years, when the two faces have worn down 1¼ inches, the tappet is replaced by a new one. A tappet rarely splits. Case-hardened tappets and cams have been tried at the Homestake mills in the expectation that the hardened working surfaces would last longer; but the tappets cracked and became useless, as they could not well be planed off. Steel tappets and cams have not been tried. It takes from six to eight hours to change the tappets and cams of one battery.

At the Homestake mills, the drop is 9 in., the number 85 per minute, and the order 1, 3, 5, 2, 4. At the Caledonia mill the height is 12 inches, the number 74, the order 1, 3, 5, 2, 4 and 1, 4, 2, 5, 3. The Caledonia mill has such a hard rock to pulverize that it is forced to have a higher drop and consequently a smaller number of drops per minute. That the Caledonia mill has two different orders of drop is quite unimportant, as they both comply with the same requisition, viz., that no stamp shall be immediately followed in falling by either of those next to it.

Cams and Cam-Shaft.—The stamps are lifted by cams fastened to a shaft which rests in boxes. These are supported by shoulders in the front of the battery posts to which they are bolted. Ten cams are keyed to one cam-shaft, which is set in motion by the cam pulley.

The cams are double-armed. They are of tough cast-iron and have the form of the involute of a circle (slightly modified at the end), the radius of which is equal to the distance from the center of the cam-shaft to the center of the stamp. The hub of the cam, which is on the off-side of the stem, is not strengthened with the wrought-iron band shrunk on it, which is often used, but is cast sufficiently thick to stand the strain. At the Homestake mills the cams have a working face 2 inches wide and 3½ inches deep. The strengthening rib, beginning at the end of the cams, gains in depth towards the hub, where it is 9½ inches deep. The hub itself is 3½ inches thick. The distance from center of hub to end of cam is 17 inches. A cam lasts three years. The cams of the Caledonia mill differ only slightly from those of the Homestake mills. The face is 2½ inches wide and 2 inches deep, the strengthening rib at the hub 10½ inches deep, the hub 3½ inches thick, the distance from center of hub to end of cam 19 inches. The cams, being made of car-wheel iron, last over four years. From ten to fifteen a year are required.

The cam-shafts are of tough wrought-iron, turned down in a lathe. They have one longitudinal key-seat. The keys with which the cams are fastened to the shaft are of steel and are hand-fitted. Wrought-iron keys lose their shape too quickly and machine-fitted keys get loose very easily; a great disadvantage. It takes ten hours to replace a broken cam-shaft, supposing that the keys have been already fitted. This ought always to be done, as the fitting of each key-seat takes about one hour. A well-appointed mill always has on hand several cam-shafts, with the necessary cams and keys ready for use. The cam-shafts of the Homestake mills were formerly made $4\frac{1}{2}$ and $4\frac{3}{4}$ in. in diameter and lasted about five years. Now the tendency is to make them stronger. The largest in use at present are $5\frac{3}{8}$ in. in diameter, and good for ten years. The distance between centers of cam-shaft and stem is $5\frac{1}{2}$ in. The cam-shaft of the Caledonia is $4\frac{3}{16}$ in. in diameter and its center is $6\frac{1}{2}$ in. distant from the center of the stem.

The cam-shaft pulleys, at the ends of the cam-shafts, vary slightly in the different mills. At the mills of the Homestake they are 6 ft. and 7 ft. 5 in., at the Caledonia mill 7 ft. 4 in. in diameter; the face is usually 15 in. wide. The pulleys are of wood. Iron could not stand the continuous shocks caused by the dropping of the stamps, and would also be too heavy. The pulley is built on a cast-iron hub with flanges, and keyed to the cam-shaft. When put in place, the shaft is made to revolve and the face of the pulley turned off to the desired form, that it may be perfectly true. The cams are lubricated with axle-grease. To prevent any of this from dropping on the apron-plates, hindering amalgamation, a curtain is stretched out beneath the cams to catch any particles of grease that these may throw off while in motion.

Crushing-Capacity.—The crushing capacity of a battery depends on the efficiency of the stamp (that is, the number of foot-pounds developed), the character of the rock under treatment and the discharging capacity of the mortar (that is, the height and size of the discharge, the character of the screen and the width of the mortar at the discharge). The Homestake stamp weighs 850 lbs. and drops 9 in. 85 times per minute; thus it develops 78,030,000 foot-pounds in twenty-four hours, crushing $4\frac{1}{2}$ tons of rock, or one ton for every 17,340,000 foot-pounds developed. The Caledonia stamp weighs 850 lbs. and drops 12 in. 74 times per minute, developing 90,576,000 foot-pounds in twenty-four hours, crushing 3.3 tons of rock, or one ton for every 27,447,272 foot-pounds developed. Thus, although

the efficiency of the Caledonia stamp is greater than that of the Homestake, it crushes less ore. There are several reasons for this. First, the character of the rock, which is much harder than that at the Homestake; second, the width of the mortar at the discharge (16 in. against 13½ in.); third, the 2-in. recess for the 8-in. amalgamated copper plate below the feed, which is absent at the Homestake mortar. From the lower discharge of the Caledonia mortar (6 in. against 10 in.) a greater crushing-capacity would naturally be expected than from the Homestake mortar. The above reasons will explain why this fails. The smallness of the Caledonia screen (258 sq. in. against 376 sq. in.) may be assumed to be counterbalanced by its character (Caledonia No. 24 wire against Homestake No. 7 slot, which corresponds to No. 30 wire). The Caledonia Co. claims that more gold is recovered by its slow method than by the more rapid one of the Homestake. This point could only be definitely settled by exact tests.

Apron-Plate, Mercury-Traps, Sluice-Boxes.—In order to save the fine particles of gold that have not been caught on the inside copper plates and to collect any amalgam and quicksilver that have passed through the screens, an amalgamated copper plate, the apron-plate, is placed in front of the mortar. The motion of the battery-water, caused by the dropping of the stamps, throws with the drop of each stamp some pulp against and through the screen. This, passing over the lip of the mortar, flows in small waves over the apron-plate. During the slight interval between these waves any heavy particles of the pulp (quicksilver, amalgam or fine gold) passing over the amalgamated plate have a chance to settle upon, adhere to and combine with it. The plate consists of one sheet of copper, nearly as wide as the discharge of the mortar, fastened with iron screws to the inclined wooden table beneath it. In all the Homestake mills (except the Deadwood and Golden Terra, where it is 12 feet long) the copper plate, $\frac{3}{16}$-in. thick, is 10 feet long, falls 2 in. to the foot, and discharges into a copper-lined trough, leading to the mercury-trap. The apron-plates of the Caledonia are 8 ft. long, 4½ ft. wide, and the copper is ⅛ in. thick. They have the same fall. The wooden table extends 4 feet beyond the end of the copper plate, at the same time narrowing to a width of 4 feet. It has a 1-in. rib down the center and is overlaid with two blankets, 5 ft. wide and 22 in. long, the upper overlapping the lower. On these the heavy sands collect. Each is washed every half hour in a tank. These blankets last six months. Carpets were tried, but they frayed and had to be thrown

out after one month. The pulp from the blankets flows into the mercury-traps, one being placed in the middle of the discharge for every plate.

The plates are of Lake copper, furnished to the mills ready for use, and do not require any annealing to make them porous. They have, however, to be flattened with wooden mallets to make them lie smoothly on the wooden table and to remove any inequalities produced during transportation. At the Homestake mills they are first scoured with sandpaper, followed by emery cloth or with tailings and a wooden block 4 inches square, or with a grindstone, until the face is perfectly bright. If necessary, the sand is moistened with a weak solution of potassium cyanide, and spots are often removed with dilute nitric acid. The surface of the pure metallic copper receives a solid coat of potassium cyanide, which is applied repeatedly as a strong solution by means of a soft paint brush. After two days the mercury is sprinkled on this cyanide coat and rubbed into the plate with a moist cloth and tailings. When the plate is thoroughly amalgamated, it is put into place and is ready for use. More than the usual amount of mercury is added to the mortar, that the new plate may become normal. This takes from two to four weeks, during which time the plates are continually discolored by copper salts. To get these into solution, potassium cyanide or ammonia is added to the battery-water. At the Caledonia mill the procedure is similar, only more care is taken to saturate the plate with mercury before it is put into use. Only the ordinary quantity of quicksilver is then added to the mortar with a new plate. In none of the mills are the plates coated with amalgam before they are put to use, nor are there any silver-plated copper plates used in the district.

The mercury-traps, through which the pulp passes on leaving the apron-plates, save amalgam and quicksilver not collected on the apron-plates. There are additional traps at the termini of the sluice-plates outside. At the Homestake mills, until about three years ago, the pulp flowed from the apron-plates directly over the sluice-plates into the waste flume. Then the traps were introduced. How important this simple contrivance is, can be seen from the fact that in the Homestake 80-stamp mill there are recovered every month, by the inside traps, 80 oz. of amalgam and 144 of quicksilver; by the outside traps, from 10 to 12 oz. of amalgam and 40 of quicksilver. These traps are emptied only once a month—on the fifteenth. If they were emptied twice a month, on clean-up days, a still better showing could perhaps be made. At the Caledonia, the traps are

emptied daily, when the apron-plates are being dressed; this is done because of the accumulation of concentrates.

At the Homestake mills, the inside trap consists of a wooden box, with copper-lined bottom, 14 in. long, 17 in. wide and 24 in. deep. It contains three sliding wrought-iron plates, parallel with its short sides. These are placed 2¼ in. apart, the center one extending to the bottom of the box, the others to 3 in. above it. The pulp flows under the first, over the middle one, and again under the third. The outside trap is larger. The wooden box is 48 in. long, 14 in. wide and 48 in. deep. It has three partitions, set 10½ in. apart, reaching from the bottom of the box up to 1½, 3 and 4 in. below the level of the inlet, the outlet being 6 in. below this. In the middle, between two wooden partitions, is let down a sliding wrought-iron plate, ⅜-in. thick, reaching to 3 in. from the bottom of the box. The Caledonia traps are much smaller, since there is one for each apron-plate.

The sluice-boxes, receiving the pulp from the inside traps, are simple wooden troughs with copper-lined bottoms. At the Homestake mills, they are from 8 to 10 ft. long, 18 in. broad and have a fall of 1 in. to the ft. The copper used is ⅛ in. thick. At the Caledonia mill they are 8 ft. long and only 8 in. broad, as less pulp passes through them. The main sluice into which they discharge is 2 ft. square.

Steam-Stamps.—The Homestake Co. has just erected an improved Ball stamp. This is a new experiment in gold-milling. The steam-stamp is successfully used for crushing native copper-ores at Lake Superior, and sulphide copper-ores at the Anaconda mine, Montana. The results of its application to gold ores may be given in a future paper.

VI. Labor in the Mills.

All the heads of the different departments are responsible to the superintendent. The mill proper is in charge of a foreman, thoroughly acquainted with every detail of work. One foreman often has the general charge of several mills, as with the Homestake and Deadwood Terra companies. Next comes the millwright, who sometimes has, as in the Homestake mill, an assistant called the pipe-fitter. The millwright combines the trades of carpenter and machinist. His duty is to make the guides, put them in place and keep them in order; to exchange cams and cam shafts, fasten any cams that have become loose, make and replace

screens, make and repair chuck-blocks, to reverse and exchange shoes and dies of crushers and to look after all the piping in the mill; in one word, to erect and keep in order every part of the mill for which any mechanical knowledge is required. One of his duties, for example, consists in lining all the shaftings and in babbitting their boxes.

In this connection might be mentioned a method of lining shafts which originated with Mr. R. Graham, the millwright of the Homestake Company, and which has proved to be quick and effective. When a shaft is to be lined, the boxes are placed so as to be approximately in line. The lower bearing, which is to receive the shaft, is loosely packed with clay and a wooden center pressed into it. This consists of a semi-cylindrical piece of dry wood having the same diameter as the shaft and about the length of the box. The wooden centers of two or more boxes are now carefully lined and the clay packed tightly around them. When in line the centers are removed, one after another, and the clay cut out crosswise in the middle of each box. The center is then replaced and the hollow space filled by pouring in babbitt. The center and remaining clay are then removed and upon the ribs of babbitt (in the middle of each box) is placed the shaft, which is now accurately in line. Finally the space left is filled with babbitt. In this way the shaft can be quickly and accurately lined, the bearings will be absolutely true and the lining from $\frac{1}{2}$ to $\frac{5}{8}$ inches thicker than the ordinary $\frac{1}{4}$-inch lining, consequently the shaft itself will last longer.

After the millwright comes the machinist. The Homestake Company having a large shop, all repairs are made there. At the Caledonia mill the foreman is also a machinist, and any extensive repairs are made by the Homestake Company.

As the mills are driven by steam, each one has two engine-men, who are responsible for their firemen. To guard against fire or any other accident, there is generally a night watchman for every mill. These men are directly responsible to the foreman.

The man who has immediate charge of the running of the mill is the head amalgamator. He also stands directly under the foreman, and is responsible for his assistants, the amalgamators, crusher-men, oilers, feeders and day-laborers, if there are any. As, in addition to running the mill, he has charge of the collecting and safe-keeping of the amalgam, he must be thoroughly trustworthy as well as capable.

The amalgamators feed quicksilver, regulate the water-supply, set

tappets, renew shoes, dies, screens and chuck-blocks, and look after the running of the battery in general. Quicksilver is fed by hand every half-hour with a little wooden spoon, similar to a mustard spoon. The quantity required in twenty-four hours ranges from ¼ to ½ lb. for every battery, according to the character of the ore, which varies a good deal. The correct amount is found out by the "feel" of the amalgam collected on the apron-plates. If this be hard and crumbly, there is danger of amalgam being carried off by the pulp, and more quicksilver is added. On the other hand, if too much quicksilver is fed into the mortar, the plates become too soft and slippery, less amalgam collects on the inner plates, and there is danger of liquid amalgam rolling off the apron-plates. Two methods of adding quicksilver are in use. At the Homestake mills, all the quicksilver required is added in the mortar, and the amalgam obtained is of medium hardness. At the Caledonia mill it is the aim, by adding only part of the quicksilver in the mortar, to make the amalgam on the inside plates as hard as may be, and to add the rest to the apron-plates, keeping these somewhat softer than those of the Homestake mills. Each management is satisfied with its own way of feeding the quicksilver, and it would be difficult to decide in favor of either method. Perhaps the fact that the Caledonia ore has, on the whole, coarser gold than the Homestake ore, may explain this difference in feeding.

The entire loss in quicksilver incurred by the Homestake management per year per stamp is 5.27 pounds, or, according to the tons of ore milled by the Homestake and Golden Star mills, 0.0044 pounds per ton of rock crushed. The entire loss incurred by the Caledonia mill is 7 pounds per year per stamp, or 0.0011 pounds per ton of rock crushed. Of course, with the hard and strongly-mineralized rock the Caledonia mill has to crush, more quicksilver will be floured per stamp than with the Homestake rock; and on account of the smaller quantity of rock crushed per stamp, less quicksilver is consumed per ton.

The amount of water required will vary according to the specific gravity of the rock, the percentage of sulphurets and the incline of the apron-plates. An excess of water will make the pulp in the mortar too thin (thus preventing an intimate contact of gold and quicksilver), will assist coarse crushing and will hinder the settling of amalgam on the apron plates. Too little water will assist amalgamation, but will hinder the pulp from passing the screens and will not carry that which has passed through them. As a rule it is

better to use too little water than an excess. The right amount will just carry the pulp evenly over the apron-plates. The Homestake mills use in twenty-four hours 1 inch of water per battery, the Caledonia mill 1¼ inches.

The setting of tappets is necessary that the stamps may always have the same height of drop, whether the shoes and dies are new or somewhat worn down. To accomplish this, the stamps are hung up, the mortar opened, the stamps then lifted up by means of an iron bar, used as a lever, and a block of wood, 1 inch higher than the desired drop, placed between shoe and die. The tappet is then loosened, allowed to fall on to the prop and again keyed fast. As the prop and the block of wood, placed between shoe and die, are both 1 inch higher than the required drop, on removing the blocks the stamps will all have the desired uniform height of drop, while the different levels at which the tappets are keyed to the stem, will indicate how much the shoe and die are worn down.

The crusher-men, in addition to looking after the grizzlies, breaking the coarse rock and feeding the crushers, have to watch for any pieces of wood and iron found amongst the ore, take it out and throw it aside. This is the only way in which the amount of chips of wood entering the mortar can be reduced to a minimum. All small pieces of wood that pass through the grizzlies are finally found in the mortar, where they are periodically removed by the amalgamators, if not previously taken out by the ore-feedermen, when the ore passes from the trough of the automatic feeder into the mortar.

The oilers have to keep all wearing iron parts of the mill lubricated and have to be especially careful not to use any excess of grease at parts where it might drop into the mortar or upon the apron plates.

The feeders attend exclusively to the regular and uniform feeding of the ore. The principle followed is to feed "low." The height of ore between shoe and die should never be more than 1 inch and as much less as is possible without the stamp beginning to pound, or, as Mr. Adams* expresses it, "let iron *almost* wear on iron."

One or two day-laborers are generally found around a large mill, because there is apt to be constant need of extra work which does not fall within the province of the regular hands.

The shifts in the mills are changed monthly. Table V. shows the amount and kind of labor in mills of 60, 80, 100 and 120 stamps:

* *Trans.*, ii. 161.

TABLE V.—*Labor and Wages.*

Class.	Caledonia,* 60 stamps.			Homestake, 80 stamps.			Father de Smet, 100 stamps.			Golden Star, 120 stamps.		
	Number employed.	Length of shift.	Wages per shift.	Number employed.	Length of shift.	Wages per shift.	Number employed.	Length of shift.	Wages per shift.	Number employed.	Length of shift.	Wages per shift.
Foreman	1	...	$6.50	⅔	...	$6.50	1	...	$5.00	⅔	...	$6.50
Millwright	1	12	4.50	1	10	4.25	1	10	4.25	1	10	4.25
Pipe-fitter	...	...	.....	⅓	10	3.50	...	...	.....	⅓	10	3.50
Enginemen	2	12	3.50	2	12	3.50	2	12	3.50	2	12	3.50
Firemen	2	12	3.25	2	12	3.00	2	12	3.00	2	12	3.00
Watchman	...	...	.....	½	12	3.00	1	12	3 00	½	12	3.00
Head Amalgamator	...	...	.....	1	10	4.00	1	12	4.00	1	10	4.00
Amalgamators	2	12	3.75	4	12	3.50	4	12	3.50	4	12	3.50
Crusher-men	1	10	3.00	5	10	3.00	6	10	3.00	6	10	3.00
Oilers	...	...	.....	2	12	3.00	2	12	3.00	2	12	3.00
Feeders	2	12	3.25	2	12	3 00	4	12	3.00	4	12	3.00
Day laborers	1	10	2.50	...	...	.....	1	10	2.50	...	...	.....
Total number	12	...	.....	20½	...	.....	25	...	.....	23½	...	.....

From the foregoing table it will be seen that the Caledonia mill employs fewer men than any of the other three mills. In comparing the 80- and 120-stamp mills of the Homestake Company, located next to each other and working on the same ore, we see that only three more men are required in the latter than in the former, although the product is half as much again. From this it can be seen that a large number of stamps is very profitable, as greatly increasing production without necessitating commensurate outlay of labor.

Daily Collecting of Amalgam and Dressing of Plates.—The amalgam which has collected on the apron-plates during the previous twenty-four hours is removed every morning when the day-shift begins. At the Homestake mills the head-amalgamator, at the Caledonia the day-amalgamator, each with an assistant, has charge of the operation. Every amalgamator has his own way of managing the details in this work, although the general outline is always the same. The method of the Golden Star mill will serve for illustration.

When the plates are to be cleaned, the water-supply of the one

* With the 20 additional stamps the Caledonia mill will require only 1 extra man, a carman, at $2.50 per shift, to transport the ore. A chute from the main ore bin to the bin of these new stamps would not have enough fall for the ore to be discharged automatically.

battery is shut off, the stamps are hung up and the splash-board removed and washed at the head of the apron plate with water through a hose. It is then placed at the lower end of the plate and the hose is turned on to the screen and apron to remove all the sands that have collected there. The plate, if normal, is now clear and bright, except for occasional spots, so-called blisters, resulting from iron and copper salts. These are removed with a scraper (a blunt, double-edged chisel). Then the two men loosen the amalgam with heavy whisk-brushes, beginning at the top and working downward. When this is done the amalgam is swept in the opposite direction and collected at the head of the apron. There it is brushed into the amalgam-scoop with a rubber scraper (a small, sharp-edged piece of belting) and emptied into a small iron receiver. After this the plates are brightened by brushing them with a whisk-broom, using tailings moistened with a dilute solution of potassium cyanide, the men working from the head of the plate downward. If necessary a little quicksilver is sprinkled on the plate from a bottle over the neck of which a piece of canvas is securely tied. After being brightened, the plates are smoothed with soft paint brushes passed transversely over them, beginning at the bottom. This finishes the operation, which requires four hours for the twenty-four plates.

The amalgam obtained is contaminated with impurities. To remove these it is placed in a Wedgewood mortar and diluted with quicksilver. The amalgamator then adds water and works the amalgam to bring all impurities to the surface. These are in part washed off (the sands) with a hose, in part removed with a sponge or wet cloth (the base-metal amalgam) until the amalgam is perfectly bright. It is then passed through a small strainer and the residual pasty amalgam is transferred to a piece of linen, where the excess of quicksilver is pressed out by wringing. The ball of hard amalgam obtained is locked up in a safe and kept until the next clean up. All the sands are returned to the battery; the waters go to waste, and the quicksilver goes back to the main stock.

The Clean-up.—Twice a month the gold amalgam adhering to the inner plates is removed, when the necessary repairs in the mill are also made. At the Caledonia mill the operations are the same at the first and fifteenth of the month and similar to those of the first of the month at the Homestake mills. At the latter the clean-up in the middle of the month differs from that at the beginning. At the first of the month the entire mortar is emptied, and shoes and dies are changed, if necessary, while on the fifteenth it is intended to

remove only the amalgam from the inside plates, and to clean up the mercury-traps.

At the Golden Star mill the clean-up at the first of the month is carried on in the following way: It begins at seven in the morning. The feeding of the battery is stopped a quarter of an hour beforehand. The stamps are made to drop slowly, so that at seven o'clock no more ore may be found in the mortar above the screen-frame. The splash-boards are removed, the stamps are hung up, the water is shut off and the engine is stopped. The mortars on one side of the mill are then opened by removing the curtains, screens and chuck-blocks. The curtains and screens are first roughly washed by playing a hose over them. They are put aside to be more carefully cleaned later on. The six chuck-blocks from the batteries facing that side of the mill which is being cleaned up are placed on two apron-plates, at each of which are four men to remove the amalgam under the supervision of the head amalgamator. This is done by scraping the plates with a chisel, when the hard amalgam drops off on the apron plate beneath. As much amalgam is removed as is possible without exposing the copper. Then quicksilver is sprinkled on the plate, to dilute the hard amalgam somewhat. This is then divided evenly over the plate and brightened by scouring with a whisk-broom and tailings, and finally smoothed with a soft paint brush. The amalgam that has dropped on the apron plate from the three chuck-blocks is collected at the head and put under lock and key by the head amalgamator. Thus the chuck-blocks of the entire mill are scraped and cleaned in four sets of six each. In the meantime another set of men scrape and wash the rim and flanges of the mortar and collect the amalgam. They also remove the plate amalgam which has settled during the past twenty-four hours. This is then also taken in charge by the head amalgamator. The dressing of the plates does not take place as yet. In order to keep them soft, a little quicksilver is sprinkled over them and evenly distributed with the brush. A third set of men begin with the work on the mortar as soon as the amalgam from the apron plate has been removed. Two small platforms are placed at its head on the wooden frame for the men to stand on. They then remove the water, still remaining in the mortar, and shovel out the sands above the dies into a heap on the apron plate (as the sands consist simply of coarse ore and do not contain any amalgam, they are returned to the battery after the dies have been put again in place). Before the die can be taken out the stamp has to be raised higher. Formerly block and

tackle were used for this purpose. Now a piece of timber is placed crosswise on the rests of the splash board, serving as a fulcrum for an iron bar with which the stamp is raised. It is kept in this position by placing a 4-inch piece of wood on the prop of the stamp and allowing the tappet to rest on it. The dies are pried up with the iron bar, lifted out and roughly cleaned. Those which are to be exchanged are taken away and piled up to be carefully scraped and washed in due time. Those that are still good (dies last six weeks) are returned to the mortar without further cleaning. After the dies have been taken out the remaining sand is shovelled out and piled up in a convenient place in the mill to be treated separately in the rocker and the pan (see later). It is rich in amalgam and contains pieces of iron that have accumulated in the mortar. Any particles of amalgam that have adhered to the rough sides of the mortar are removed and added to the sands. The dies are now put in place again. If new shoes are required they are placed on top of the dies, with the wooden collar slipped over the shank. Then the recesses for the chuck-block, screen-frame, etc., are cleaned by directing a hose upon them, and these are put in place, the screens having first been cleaned in a wooden box with brush and water. When the chuck-block is in place, the sands first removed are shovelled in to fill the bottom of the mortar up to the top of the dies. The drop of the stamp has now to be regulated. If new shoes are used, the wooden block, 1 inch higher than the drop, is placed on the shoulder of the shoe and the stamp let down until the head rests on the block. If the shoe has not been replaced, the block stands upon the die. In both cases the keys of the tappets are loosened, these are allowed to fall on to the prop and are then keyed up again. The apron-plates are now dressed in the usual way. Any amalgam adhering to the small sluices, leading to the mercury-traps, and to the sluice-boxes is removed and these are brightened in the same manner as the apron-plate. When the engine has been started up, the stamps that have new shoes are first allowed to drop several times until the sound and the vibration, felt when touching the tappet, show that the shoe is thoroughly fastened to the head. The splash-boards are put back in place, some ore is fed into the mortar, the water is turned on and the stamps of one battery after another are let down from the props. Special care has to be taken by the feeders to regulate the ore supply, as the mortars are empty above the dies when the mill starts up.

In cleaning up a mill, all the hands have to take part, the night shift working six hours extra.

This detailed description of the clean up of a 120-stamp mill has been given, because it seemed important to show how it is possible to accomplish it in the short space of seven hours without requiring any outside help. When it is remembered that more than a day was formerly consumed in this work, it will be seen how admirably the operations are now systematized and worked one into another.

When the clean-up of the mill is over and the stamps are once more in operation, the sands that were shovelled out from the bottom of the mortar have to be worked up and the amalgam has to be cleaned, so as to be ready for the retort. Two crusher-men are detailed to clean up the sands. These contain bits of iron that have come from the tools used in the mine or the crushing machinery of the mill, and also pieces of amalgam. The sands are first treated in a rocker. Any coarse pieces of iron are picked up and collected in a separate heap. When the sands have been rocked for a little while and the hose has been played on them, the residue on the hopper is broken as fine as possible with a wooden mallet. The products obtained by rocking are the coarse particles remaining finally in the hopper. These are washed in a coarse screen over the clean-up pan, and any amalgam remaining on the screen is picked out and thrown into the pan. The sands go back to the battery. The heavy sands that collect on the curtain and riffle are taken up in a bucket to be worked in the pan. Those settling in the sluice which conducts the slimes to the waste flume are shovelled out and returned to the battery.

There remains the purifying of the amalgam collected from the chuck-blocks, apron-plates, sluices leading from the mercury traps, mortars, old shoes, dies and screens. It is charged with water into the clean-up pan (5 ft. in diameter, the muller making 30 revolutions per minute) and from 600 to 700 lbs. of quicksilver are added. It takes about three hours to clean in the pan all the by-products that contain any amalgam. When this is all collected and the water above it is somewhat clear, the muller is raised with block and tackle and the entire contents of the pan are emptied through the lowest discharge-opening into a square box which overflows into the tailings-discharge box. The muller and bottom of pan are cleaned with brush and water. The liquid amalgam in the first box is now freed from water and passed through a strainer. The resulting pasty amalgam is removed, and freed from the excess of quicksilver by wringing it in canvas bags in water. The balls of hard amalgam remaining in the bags contain about 38 per cent. of gold. The

quicksilver collected beneath the strainer goes back to the main stock; that obtained by squeezing the pasty amalgam is first purified by adding some nitric acid, stirring it, and then washing it with water.

The clean-up on the fifteenth of the month is much simpler than that on the first, as only the chuck-blocks are taken out and the mortar is left intact, except, of course, when any break has occurred in shoe or die. On the fifteenth, however, the mercury-traps are cleaned. Their contents go simply into the pan and are worked with the rest of the products containing amalgam. This clean-up lasts five hours—that is, two hours less than the one on the first of the month.

The cleaning up of old iron and wood chips, which is done once a year, still remains to be mentioned in this connection. The pieces of iron that are picked out from the sands in the bottom of the mortar are first scraped, to remove any amalgam adhering to them. They are then thrown out upon an enclosed heap in the yard and left there to be corroded by atmospheric action. The rusting is hastened by adding some salt to the heap at various times. Once a year, the iron that has entirely fallen to pieces is charged with quicksilver into the pan and its gold extracted. The chips of wood, which float on the top of the water in the battery and often clog the screens, are repeatedly removed and collected in a box in one of the window recesses. They are then emptied on a heap in the yard, and once a year are set on fire. The ashes are collected and amalgamated in the pan. In this way are saved every year from 16 to 18 pounds of amalgam from the two mills of the Homestake Company (200 stamps).

Retorting and Melting.—The quicksilver still retained by the hard amalgam is removed by retorting. Both bulb retorts and cylindrical ones are used. They are all of cast-iron. The cylindrical retort of the Homestake Company is 12 in. in diameter and 3 ft. long, and holds about 1000 lbs. of amalgam. The usual charge of 500 lbs. is retorted in about 6 hours with ¼ cord of wood. The loss of quicksilver is nominal. The retort-metal amounts to 38 and 40 per cent. of the original charge. At the Caledonia mill it is only 33 per cent., less care being taken in cleaning the amalgam; and at the Deadwood-Terra mills it is often only 25 per cent., by reason of the fine condition of the gold, which requires more quicksilver to form amalgam.

The retort-metal is now melted into bars. At the Homestake mill, two kinds of moulds are used, the 1500-ounce silver mould, 5 by 5

by 11½ inches, and the 700-ounce mould, 3½ by 4 by 9½ inches. The bars are cast from 3 to 4 inches thick and weigh from 1000 to 1400 ounces. It takes about four hours to melt four 1400-ounce bars and the crucible lasts for from eight to twelve charges.

The loss in melting with Homestake retorted bullion is only 1.5 per cent. and the average composition of the bars is 820 gold, 165 silver, 15 base metal. The loss of the Caledonia bullion is greater (7 per cent.), as less care is taken in cleaning the amalgam. The average composition of its bullion is 798 gold, 182 silver and 20 base metal.

The bullion is sampled, weighed, assayed and shipped.

VII. Working-results.

The Homestake and Golden Star mills together produced, according to the report of the Homestake Company, from June, 1887, to June, 1888, $895,822.37 worth of gold from 243,355 tons of ore, which corresponds to a yield in free gold of $3.68 per ton. It is claimed that 85 per cent. of the free gold is saved. According to this the ore would run in free gold $4.33 per ton. The total value of the ore varies from $5 to $10 per ton; the amount of concentrates does not exceed 3 per cent. Their value, as shown by experiments in the pan, is $24 per ton, although judging from the average assay of the tailings, ($1.50 per ton) a much higher figure would be expected.

Two sets of experiments were made in the spring of 1885 by Dr. R. Goering on the Homestake and Golden Star tailings, the first when the mills had no mercury-traps, the second after these were introduced. Up to that time there had been collected by means of blankets in a separate building (the blanket house) 1124 tons of concentrates which assayed $35.16 per ton. These, panned down, gave 20.5 per cent cleaner concentrates, assaying $40.18 per ton. When amalgamated in the pan they yielded 56.9 per cent. of their total value, the remaining pure pyrites assaying $17.08 per ton. This shows that the gold recovered must have been either enclosed in quartz or rusty, or that amalgam had been carried off with the tailings and was recovered by the blankets. When in consequence of these tests the mercury traps were introduced, the loss was reduced. The concentrates then saved assayed $27.63 per ton. They gave up 92 per cent. of their gold in the pan, but the pure pyrites still assayed $12.26 per ton, showing that while the loss had been diminished, it had not been stopped. The con-

centration of tailings by blankets, being too expensive, was given up.

From the other mills under the Homestake management no official reports are made public; therefore no data can be given. It may, however, be stated that the tailings from Highland ore average $1 per ton, those from Deadwood-Terra ore $0.50 per ton, and seldom exceed $0.75. It would be interesting to find how small the amount of concentrates is and how high these would run.

One point still remains to be considered in regard to working-results, that is the fineness to which it is necessary to crush the ore. Dr. R. Goering made tests to find a relation between size and assay value of tailings. Samples were taken hourly for a length of time and the sands obtained dried, weighed and screened through different sieves. The figures are given below:

TABLE VI.—*Relation between Size and Assay-Value.*

Per cent. in weight.	Passing through screens, No.	Remaining on screens, No.	Assay value per ton.
94.07	50	...	$1.25
2.53	50	40	2.13
3.40	40	...	2.76

They show that the loss in the tailings increases rapidly, if the screens are allowed to remain too long in the mortar. This is still better shown by another series of experiments (December, 1888) on tailings, running $2 per ton, that were screened through a No. 30 mesh sieve. Of these, 6 per cent., that did not pass through the screen, assayed as high as $5.02 per ton. The result is, as has been said before, that the heavy Russia-iron slot screens have to be exchanged every fortnight.

The Caledonia mill crushed, from May, 1887, to May, 1888, 73,425 tons of rock and produced $295,816.25 worth of bullion, equalling a yield of $4.02 per ton in free gold. Figures corresponding to those given in the Homestake Company could not be obtained. It may, however, be stated that the blanket concentrates of the Caledonia, amalgamated raw in the pan, yield a product of pure pyrites, assaying $90 to the ton, and that the tailings from the blankets, when panned, yield concentrates worth from $7 to $9 to the ton.

TABLE VII.—*Cost of Milling in* 1887–8 *at Homestake and Golden Star Mills.*

Items.	Homestake Mill. (96,790 tons treated.)		Golden Star Mill. (146,565 tons treated.)	
	Amount expended.	Cost per ton.	Amount expended.	Cost per ton.
Labor	$24,789.90	$0.2561	$31,338.60	$0.2138
Supplies	1,263.71	0.0130	1,149.65	0.0079
Water	16,731.37	0.1729	25,097.10	0.1712
Wood	26,773.50	0.2766	40,156.50	0.2739
Machinery	8,924 06	0.0922	17,884.96	0.1220
Oil	1,052.69	0.0109	1,233.21	0.0084
Candles	156.00	0.0016	214.50	0.0014
Quicksilver	1,001.25	0.0103	3,701.05	0.0252
Lumber	684.06	0.0070	799.16	0.0054
Total	$81,376.54	$0.8406	$121,574.73	$0.8292

In comparing the cost of milling in the 80- and 120-stamp mills, it seems strange that there should be so little difference. Special causes must have affected this year's work, for the cost per ton in 1880* was $1.3018 and $0.7770 respectively.

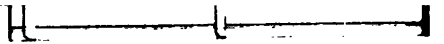
The cost of milling at the Caledonia mill for 1887–88 was $0.87 per ton, $\frac{1}{3}$ going to labor and $\frac{2}{3}$ to material. This is a low figure, if we consider that the mill has only 60 stamps and that the ore treated is very hard in comparison with any Homestake rock.

VIII. CONCLUSIONS.

Two striking features are presented by the foregoing review; the simplicity and effectiveness of the methods by which the free gold is extracted; and the evident waste of sulphurets in the tailings. These sulphurets appear according to the figures given above (3 per cent., assaying $24 per ton) to be quite rich enough to repay working.

A simple and cheap method of concentration, for the large quantities to be treated, would probably be as follows: The pulp, after passing over a series of spitzlutten, which would sort out the coarse sands, would overflow into a series of spitzkasten, and the overflow of these would be waste. The coarse sands, sorted out by the spitzlutten, would contain, according to experiments made in 1885 on the large heap of concentrates, free gold, which would be recovered by crushing them wet in rolls or a Chilian mill, allowing the pulp

* *Tenth Census of the United States*, vol. xiii., p. 280.



to flow over amalgamated plates, and then pass over a separate series of classifiers or else go back to the main system. The graded pulp, obtained by the different spitzkasten, would be separated on continuous round tables, into pure concentrates, middlings and waste.

The middlings would be worked separately, or be pumped back to the main system of spitzkasten. The cost of concentrating the tailings would probably not exceed $1 per ton of sulphurets.

The method of working these concentrates would be that of barrel-chlorination, using bleaching powder and sulphuric acid to generate the chlorine. In a paper read before the Institute at the Buffalo meeting, October, 1888, Prof. William B. Phillips says the cost of dead-roasting on a 12-ft. revolving hearth, at the Phœnix mines, North Carolina, is $2.03 and $2.18 per ton. A combination of two systems of furnaces would probably accomplish this dead-roasting at a proportionately cheaper rate at the mills, where wood is $6 a cord and labor $3 and $3.50 a day. The Spense automatic desulphurizing roaster would do the preliminary roasting cheaply, and the revolving hearth would effectually dead-roast large quantities of ore, whose sulphur had been nearly all eliminated. Prof. Phillips further gives the cost of chlorinating at $2.66 per ton of concentrates. This figure would have to be considerably increased for the Homestake and other mills. The total cost would probably be not over $8 for the ton of concentrates.

In closing this paper, the writer desires to express his thanks to the management of the Homestake and Caledonia mills for their courtesy in allowing him the run of the works and to Dr. R. Goering for his valuable assistance in obtaining detailed information. Also to Messrs. R. Graham and Chas. Ruth, for their uniform kindness in answering questions.

Discussion.

Auguste Mathez, of Central City, Colorado (communication to the Secretary): A simple and cheap method of concentration is in use in Gilpin county, Colorado, where the battery-tailings from the copper-plates are run over Bertenshaw and McFarland bumping-tables, which have done close work. The ores of Gilpin county are chiefly sulphurets of iron and copper with a quartzose gangue. On this class of ore the mills crush on an average 5 tons of rock per battery of five stamps in 24 hours. One bumping-table is able to handle the battery-tailings from 10 stamps. The tables

can be put up for about $250 each. One attendant is able to do the work on five tables.

The concentrates are sold to smelters, $9 per ton being deducted for freight and treatment; but the barrel-chlorination suggested by Mr. Hofman seems applicable to the conditions prevailing in Gilpin County, and if so, the cost per ton of concentrates could probably be reduced to $8, or less.

THE DETERMINATION OF SILICON IN FERRO-SILICONS; ITS OCCURRENCE IN ALUMINUM AS GRAPHITOIDAL SILICON; AND A STUDY OF ITS REACTIONS WITH ALKALINE CARBONATES.

BY HENRY J. WILLIAMS, ST. LOUIS, MO.

(New York Meeting, February, 1889.)

THE main difficulty in the determination of silicon in pig-irons containing very high percentages of that element, has been due to their almost complete insolubility in acids, or mixtures of acids. The object of the present paper, which presents no new principle, is to suggest a method of determination which has repeatedly given accurate results, and which obviates completely the tediousness and uncertainty of the usual slow solution in acids.

This method has been tried with pig-irons containing from 7 to 14 per cent. of combined silicon, and properly called ferro-silicons. When reduced to an impalpable powder in the agate mortar, an 11 per cent. ferro-silicon was found to resist the action of all acids, or mixtures of acids, with the exception of sulphuric acid of about 1.25 specific gravity. But the action of this acid, even during long continued heating, was very slow, 72 hours being necessary to completely dissolve half a gramme of the metal.

The complete decomposition of 1 gramme of the metal may, on the other hand, readily be effected in a few minutes by fusion with sodium carbonate in a covered platinum crucible over a blast-lamp —the idea being to dissolve the silicon, leaving the iron in condition to be readily attacked by acids. The silicon dissolves as soluble sodium silicate, and metallic iron remains in a very spongy and finely divided condition. The addition of a little hot water

and strong hydrochloric acid gives a perfectly clear solution, showing the complete decomposition of the ferro-silicon. The silica obtained by ignition, after the usual evaporation to dryness in a platinum dish, is absolutely snow-white.

Fifteen or twenty minutes over the blast-lamp are quite sufficient to effect the solution of the silicon, and final results may easily be obtained within three hours.

The following analyses show some results actually obtained:

Percentage of Silicon.

Sample.	I.	II.	III.
No. 1, ...	10.92	11.01	10.98
No. 2, ...	10.73	10.75	10.73
No. 3, ...	7.42	7.48	7.46
No. 4, ...	13.41	13.37	
No. 5, ...	13.77	13.74	
No. 6, ...	13.96	14.01	

Some curious phenomena occur during the fusion, which seem worthy of note.

As soon as the sodium carbonate is thoroughly melted and the heat reaches its maximum, the action is very violent, and bubbles of gas rise to the surface and there burst into flame. The gas given off is carbonic oxide. Dr. Thos. M. Drown noticed this several years ago, while making some experiments of a similar nature with graphitic pig-irons. His results were communicated to the Institute (*Trans.*, vii. 146). Others also have mentioned it. My experiments, however, were made with very high silicon and comparatively low carbon, and seemed to give the same results.

Apparently the two following reactions take place:

(1) $Na_2OCO_2 + C = Na_2O + 2CO$.

(2) $2\ Na_2OCO_2 + Si = Na_2SiO_3 + Na_2O + 2CO$.

The carbon and the silicon seem to exert an entirely similar reducing action, for every molecule of sodium carbonate yields to them an atom of oxygen and sets free a molecule of carbonic oxide gas.

While endeavoring to prove that the silicon acts exactly like the carbon during this reduction, I vainly sought for an iron entirely free from foreign elements, particularly carbon, but containing a considerable amount of silicon. Failing in this, I succeeded in finding the desired condition realized in the chemically pure aluminum of trade, which, in spite of its name, rarely contains less than 3 or

4 per cent. of silicon. But a large part of the silicon in aluminum seems invariably to be present as an allotropic modification of that metalloid which crystallizes in fine glistening black plates resembling some forms of graphite.

This is evidently the graphitoidal form of silicon which Deville has mentioned in connection with aluminum, but which has not yet been isolated or found to exist in iron. These plates are insoluble in the mineral acids and resist their oxidizing action.

The determination of silicon in aluminum by the usual method—that of solution in acid and evaporation to dryness, gives results that are necessarily too low, as the bluish-black residue obtained after ignition, which is usually weighed as silica, consists not only of silica but also of unoxidized silicon in the graphitoidal condition. I have generally found the results obtained in this way to be about 30 per cent. too low, while the metal made abroad, which contains, it is claimed, 96 per cent. of aluminum, really carries less than 94 per cent.

In order to secure correct results, it is necessary to fuse this residue with sodium carbonate, and to evaporate the hydrochloric acid solution of the fused mass once more to dryness. The silica obtained by ignition will then be white and completely oxidized.

During this fusion, carbonic oxide gas is abundantly given off, showing that the sodium carbonate is undergoing decomposition, and the flames that have already been mentioned occur, thus proving that silicon acts precisely like carbon in setting this gas free from the alkaline carbonates.

A fusion was then tried with a wrought-iron almost free from silicon or other elements, and containing no carbon whatever, in order to ascertain whether, at a red heat, metallic iron would decompose sodium carbonate. Two grammes of coarse drillings were heated for two hours in a platinum crucible with 10 grammes of sodium carbonate. An almost imperceptible action was noticed; but it soon ceased, and gas was at no time evolved in sufficient quantity to take fire.

Upon examination, the silicon in the borings, which had been originally 0.18 per cent., was now found by analysis to be 0.03 per cent.

Numerous experiments have shown that the removal of silicon from a ferro-silicon, or any other kind of iron containing it, by fusion with an alkaline carbonate, is always accompanied by an evolution of carbonic oxide gas, and that the removal is complete as soon as this evolution of gas ceases. It seems, therefore, reasonable to ascribe the slight action just mentioned to the removal of the silicon

from the wrought iron, which was shown to have taken place, further action ceasing when this point was reached. But the borings, which had been bright before the fusion, were now found to be coal-black, and to have become entirely insoluble in dilute or strong nitric acid, even upon heating, while the original drillings readily dissolved in the cold acid. Apparently, the removal of the silicon was accompanied by a superficial oxidation of the borings, for a coating of oxides of iron would resist the action of nitric acid. A few drops of hydrochloric acid, however, were sufficient to effect a solution. The black color of the coating and the conditions under which it was formed give some indication of its nature; it probably consists of a mixture of the protoxide and sesquioxide of iron; in other words, of the magnetic oxide, Fe_3O_4.

Pure metallic iron at a red heat may, therefore, be said to have no action in decomposing sodium carbonate. Iron, however, which contains either carbon or silicon, or both, gives off carbonic oxide gas during the fusion with the alkaline carbonates, and this burns to carbonic dioxide as soon as it reaches the air. Furthermore, the giving up of oxygen to the carbon and silicon is not confined entirely to those elements, for the superficial oxidation of the drillings of wrought iron containing a little silicon indicates very plainly that sodium carbonate which is undergoing decomposition may act as an oxidizer for iron at a red heat. This action is similar to that of carbonic dioxide in the blast-furnace, where, under certain conditions, this gas is an oxidizer.

Borax, at a fusing temperature, was found capable of decomposing the ferro-silicons, and carbonic oxide was evolved during the fusion; but in the determination of silicon, sodium carbonate possesses marked advantages over it.

The decomposition by fusion with bisulphate of potassium, however, with the idea of leaving the silicon insoluble, gave uncertain results and was frequently incomplete.

The evolution of carbonic oxide from the decomposition of an alkaline carbonate during its fusion with a ferro-silicon does not occur if an oxidizing agent, such as potassium nitrate, is present. Here, as both the silicon and the carbon find in the nitre plenty of oxygen with which to combine, it is not necessary for them to reduce the alkaline carbonate in order to secure it, and hence there is no evolution of carbonic oxide gas, nor are there any flames.

SOAPING GEYSERS.

BY ARNOLD HAGUE, WASHINGTON, D. C.

(New York Meeting, February, 1889.)

AT the Buffalo meeting, October, 1888, Dr. Raymond presented a paper entitled: "Soaping Geysers" (p. 449 of the present volume), in which he called attention to the use of soap by tourists to cause eruptions of several of the well-known geysers in the Yellowstone Park. Incorporated in this paper appears a communication received from me, written from camp in the Park, in reply to some inquiries on the subject. The letter discussed somewhat briefly the means employed by visitors to the Park to hasten the eruptions from hot springs and reservoirs of hot water, which remain dormant for days, or even weeks or months, at a temperature near the boiling-point, without any display of geyser-action. As the paper has called forth considerable comment, I desire to elucidate one or two points in relation to the temperature of the springs, and to answer some inquiries about the composition of the thermal waters.

In the summer of 1885, a Chinaman, employed as a laundryman for the accommodation of the tourists at the Upper Geyser Basin, accidently discovered, much to his amazement, that soap thrown into the spring from which he was accustomed to draw his supply of water produced an eruption in every way similar to the actual workings of a geyser. Tourists with limited time at their command, who had travelled thousands of miles to look upon the wonders of the Yellowstone, soon fell into the way of coaxing the laundryman's spring into action, to partly compensate them for their sore disappointment in witnessing only the periodical eruptions of Old Faithful. Successful attempts upon this spring soon led to various endeavors to accelerate action in the dormant and more famous geysers. In a short time, so popular became the desire to stimulate geysers in this way, that the Park authorities were compelled to enforce rigidly the rule against throwing objects of any kind into the springs.

In connection with a thorough investigation of the thermal waters of the Yellowstone Park and the phenomena of the geysers, I undertook a number of experiments to ascertain the action of soap upon the waters and to determine, if possible, those physical conditions of various pools and reservoirs which permitted the hastening of an

eruption by the employment of any artificial methods. This investigation, conducted from time to time, as opportunity offered, throughout the field-season of 1885, included experiments upon the geysers and hot springs of the Upper, Lower and Norris geyser basins. The results proved, beyond all question, that geyser-action could be forced in a number of ways, but most conveniently by the application of soap. The greater part of the more powerful geysers undergo no perceptible change with a moderate use of soap, although several of them may, under favorable physical conditions, be thrown at times into violent agitation. In most of the experiments, Lewis's concentrated lye, put up in one-half pound cans for laundry purposes, was employed. Each package furnished a strong alkali, equivalent to several bars of soap. In this form alkali is more easily handled than in bars of soap, more especially where it is required to produce a viscous fluid in the larger reservoirs; and, in conducting a series of experiments for comparative purposes, it seemed best, in most instances, to employ the same agent to bring about the desired results.

Old Faithful, the model geyser of the Park, exhibits such marked regularity in its workings that attempts to hasten its action appear futile. The interval between eruptions is about 65 minutes, and rarely exceed the extreme limits of 57 and 72 minutes. After an eruption of Old Faithful, the reservoir fills up gradually; the water steadily increases in temperature; and conditions favorable to another eruption are produced under circumstances precisely similar to those which have brought about the displays for the past eighteen years, or as far back as we have authentic records. The few experiments which have been made upon Old Faithful are insufficient to afford any results bearing on the question; but it seems probable that soon after the water attains the necessary temperature an eruption takes place.

Of all the powerful geysers in the Park, the Bee-Hive offers the most favorable conditions for producing an eruption by artificial means, all the more striking because the natural displays are so fitful that they cannot be predicted with any degree of certainty. Observations, extending over a period of several years, have failed to determine any established law of periodicity for the Bee-Hive, even for three or four consecutive months; although they indicate that some relationship may exist between its display and those of the famous Giantess. Frequently the Bee-Hive will play several times a day and then become dormant, showing no signs of activity for

weeks and months, although the water may stand above the boiling-point the greater part of the time. The name Bee-Hive was suggested by the symmetry of the cone built around the vent. It rises about 4 feet above the sloping mound of geyserite, and, in cross-section, measures about 3 feet at the top, while at the bottom of the cone the vent is less than 10 inches in width. From the top of this narrow vent it is only possible to sink a weight 17 feet before striking a projecting ledge, which interferes with all examination of the ground below. The constant boiling and bubbling of the water, the irregularity of its action, and the convenient location of the geyser, within an easy walk from the hotel, make attempts to accelerate the eruptions of the Bee-Hive most attractive to tourists.

In most instances such efforts are futile; yet success does so frequently reward the astonished traveller that, unless the geyser were carefully watched by the authorities, attempts would be made daily throughout the season. If the conditions are favorable to an eruption, it usually takes place in from 10 to 25 minutes after the addition of laundry soap or lye.. It is doubtful if more than two eruptions of the Bee-Hive have ever been produced on the same day by artificial means, although I know of no reason, based upon the structure of the geyser, why more displays might not be obtained; for the reservoir and vent fill up with boiling water very rapidly after each eruption.

Athough the Giantess is situated only 400 feet from the Bee-Hive, these two differ in surface and underground structure, and mode of action, as widely as any two of the more prominent geysers of the Park. Around the Giantess no cone or mound has formed. The broad basin is only partially rimmed in by a narrow fringe of siliceous sinter, rising above and extending out over the deep blue water. At the surface, this basin measures about 15 to 20 feet in width by 20 to 30 feet in length. It has a funnel-shaped cauldron, 30 feet in depth, ending in a vertical vent or neek, 12 feet deep, through which a sounding-lead may be dropped into a second reservoir, meeting a projecting ledge or obstruction of some kind, 61 feet below the surface. After an outburst of the Giantess, the basin, which has been completely emptied of its water, gradually fills again to the top; and, for days before another eruption, a steady stream of hot water overflows the brim. The intervals between the eruptions of the Giantess vary from 12 to 20 days, and the displays last several hours, being unsurpassed for violence and grandeur by any geyser in the Upper Basin. Artificial means have never been successful in

bringing this geyser into action, although, for days before an eruption, it is an easy matter to cause an agitation of the water by throwing into the basin small pieces of sinter, or to produce a boiling on the surface, lasting several minutes, by simply stirring the water with a stick.

The Giant, one of the most violent of the geysers in the Upper Basin, more closely resembles the Bee-Hive than any other of those along the Firehole river. It has built up a cone 10 feet in height, one side of which has been partly broken down by some eruption more violent than any witnessed at the present day. Through this notched side, steam and broken jets of water are constantly emitted, and on this account but little examination has been made of the underground reservoirs and vents. The Giant is fitful in its action, at times playing with considerable regularity every 14 days, and at other times lying dormant for nearly a year. I have no positive knowledge that an eruption of the Giant has ever been produced by any other than natural causes. At the time of my experiments no eruption of the Giant had taken place for several months, although the water was constantly agitated, so much so that it was quite impossible to examine the vent with any satisfactory results. The only effect produced by the application of lye was additional height to the column of water thrown out and a decided increase in the thumping and violence of the boiling.

In the Lower Basin, the Fountain has been more carefully studied than the other geysers; and, its action and periodicity of eruptions having been fairly well ascertained, it afforded the most favorable conditions for observing the action of soap and lye upon the waters. In its general structure the Fountain belongs to the type of the Giantess, having a funnel-shaped cauldron which, long before an eruption, overflows into an adjoining basin. At the time of my experiments upon the Fountain, the intervals between eruptions lasted about four hours. This interval allowed sufficient time to note any changes which might take place. My own experiments with lye yielded no positive results; although it seemed highly probable that action might be hastened by the application of soap or lye just before the time for an eruption, or when, for some cause, the eruption was overdue. I preferred to make the attempt to bring about an explosion before the usual time, only waiting until the water in the pool had nearly reached the boiling-point. All experiments failed. The previous year, when wishing to produce action for the purpose of photography, I was enabled to accomplish the desired result by

vigorously stirring, with a slender pole, the water near the top of the vent connecting with the lower reservoir. In this instance, it should be said, the usual interval of time between eruptions had long since passed; the geyser was, so far as time was concerned, a half-hour overdue. My opinion now is that the experiments with lye failed because the temperature had scarcely reached the boiling-point.

The Monarch, in the Norris Basin, is quite unlike those already described, and affords evidence of being a much newer geyser. It is formed by two convergent fissures, on the line of a narrow seam in the rhyolite, probably coming together below the surface. The main vent measures about 20 feet in length and, at the surface, 3 feet in width. But slight incrustation is found around the vent, the conditions not being very favorable to deposition. In this narrow fissure the water, which ordinarily stands about 15 feet below the surface, constantly surges and boils, except immediately after an eruption. The intervals between eruptions vary somewhat from year to year; but at the time of these experiments the action was fairly regular, the geyser playing every four hours. I was successful in obtaining an eruption quite equal to the natural displays, which throw a column of water 50 feet into the air. Here at the Monarch there is no surface reservoir, and the narrow fissure, filled with loose blocks of rocks around which the water is in constant agitation, prevents all measurements of depth.

The results of the many experiments, not only upon active geysers, but upon a large number of hot springs, determine fairly well the essential conditions which render it possible to bring about geyser-action by artificial means. Negative results are frequently as valuable for this inquiry as experiments yielding imposing displays.

Outside of a few exceptional instances, which could not be repeated, and in which action was probably only anticipated by a few minutes in time, geyser eruptions produced by soap or alkali appear to demand two essential requirements: First, the surface-cauldron or reservoir should hold but a small amount of water, exposing only a limited area to the atmosphere; second, the water should stand at or above the boiling-point of water for the altitude of the geyser basin above sea-level. The principal factor which makes it possible to cause an eruption artificially is, I think, the superheated and unstable condition of the surface-waters. Many of the geysers and hot springs present the singular phenomena of pools of water heated above the theoretical boiling-point, and, unless disturbed, frequently remain so for many days without exhibiting any signs of ebullition.

It may not be easy to describe accurately these superheated waters; but any one who has studied the hot springs and pools in the Park and carefully noted the temperatures, quickly learns to recognize the peculiar appearance of these basins when heated above the boiling-point. They look as if they were "ready to boil," except that the surface remains placid, only interrupted by numerous steam-bubbles, rising through the water from below and bursting quietly upon reaching the surface.

Marcet, the French physicist, has specially investigated the phenomena of superheated waters, and has succeeded in attaining a temperature of 105° C. before ebullition. Superheated waters in nature, however, appear to have been scarcely recognized, except during the progress of the work in the Yellowstone Park, in connection with a study of the geysers. The altitudes of the geyser basins above sea-level have been ascertained by long series of barometric readings, continued through several seasons. In conducting a series of observations upon the boiling-points of the thermal waters in the Park, Dr. William Hallock, who had charge of this special investigation, determined the theoretical boiling-point by noting the mean daily readings of the mercurial column. The exact boiling-point of a pure surface-water, obtained from a neighboring mountain stream and the boiling-point of the thermal waters from the springs, were determined from actual experiments by heating over a fire, employing every possible precaution to avoid sources of error. Surface-waters and deep-seated mineral waters gave the same results, and coincided with the calculated boiling-point at this altitude. Hundreds of observations have been carefully taken where the waters in the active and running springs boiled at temperatures between 198° and 199° Fahr.

As will be shown later in this paper, the thermal waters are solutions of mineral matter too dilute to be affected to any appreciable extent as regards their boiling-point by their dissolved contents. The theoretical boiling-point for the springs and pools in the Upper Geyser Basin may be taken at 92.5° C. (198.5 Fahr.). In many of the large cauldrons, where the water remains quiet, a temperature has been recorded of 94° C. (201.2° Fahr.) without the usual phenomena of boiling. This gives a body of superheated water, with a temperature at the surface 1.5° C. (2.7° Fahr.) above the point necessary to produce explosive action. Thermometers plunged into the basins show slightly varying temperatures, dependent upon their position in the basin. They indicate the existence of numerous

currents, and a very unstable equilibrium of the heated waters, which are liable, under slight changes, to burst forth with more or less violence. It is under these conditions that geyser-action can be accelerated by artificial means. If into one of these superheated basins a handful of sinter pebbles be thrown, or the surface of the water be agitated by the rapid motion of a stick or cane, or even by lashing with a rope, a liberation of steam ensues. This is liable to be followed by a long boiling of the water in the pool, which in turn may lead to geyser-action. There is some reason to believe that, at least in one instance, an eruption has been brought about by a violent but temporary gust of wind, which either ruffled the water or disturbed the equilibrium of the pool, and changed momentarily the atmospheric pressure.

In Iceland travellers have long been accustomed to throw into the geysers turf and soft earth from the bogs and meadows which abound in the neighborhood, the effect produced being much the same as that of sinter pebbles and gravel upon the geysers in the National Park. So well was this understood that at one time a peasant living near the Iceland locality kept a shovel solely for the accommodation of those visiting the geysers.

In my letter to Dr. Raymond I mention the curious fact that the laundryman's spring, now known as the Chinaman, in which geyser-action may most easily be produced by artificial means, has never been regarded by the Geological Survey as anything but a hot spring, and no one has ever seen it in action without the application of soap, except in one instance, when it was made to play to a height of twenty feet after stirring it vigorously with a pine bough for nearly ten minutes. In our records it is simply known as a spring.

If soap or lye is thrown into most of the small pools, a viscous fluid is formed; and viscosity is, I think, the principal cause in hastening geyser-action. Viscosity must tend to the retention of steam within the basin, and, as in the case of the superheated waters, where the temperature stands at or above the boiling-point, explosive liberation must follow. All alkaline solutions, whether in the laboratory or in nature, exhibit, by reason of this viscosity, a tendency to bump and boil irregularly. Viscosity in these hot springs must also tend to the formation of bubbles and foam when the steam rises to the surface, and this in turn aids to bring about the explosive action, followed by a relief of pressure, and thus to hasten the final and more powerful display. Of course relief of pressure of the superincumbent waters upon the column of water below the sur-

face basin is essential to all eruptive action. These conditions, it seems to me, are purely physical. Undoubtedly the fatty substances contained in soap aid the alkali in rendering the water viscous. On the other hand, when concentrated lye is used it acts with greater energy, and furnishes a viscous fluid where soap would yield only surface suds, insufficient to accomplish any phenomenal display.

It is well known that saturated solutions of mineral substances raise the boiling-point very considerably, the temperature having been determined for many of the alkaline salts. In general, I believe the boiling-point increases in proportion to the amount of salt held in solution. Actual tests have shown that the normal boiling-point of siliceous waters in the Park does not differ appreciably from the ordinary surface-waters, mainly, I suppose, because they are extremely dilute solutions.

The amount of lye required to produce a sufficiently viscous condition of the waters increases but slightly the percentage of mineral matter held in solution.

All the waters of the principal geyser-basins present the closest resemblance in chemical composition, and, for the purposes of this paper, may be considered as identical in their constituents. They have a common origin, being, for the most part, surface-waters which have percolated downward for a sufficient distance to come in contact with large volumes of steam ascending from still greater depths. The mineral contents of the hot springs are mainly derived from the acid lavas of the Park plateau, as the result of the action of the ascending steam and superheated waters upon the rocks below. These thermal waters are essentially siliceous alkaline waters, carrying the same constituents in somewhat varying quantities, but always dilute solutions, never exceeding two grammes of mineral matter per kilogramme of water. When cold they are potable waters, for the most part slightly alkaline to the taste, and probably wholesome enough, unless taken daily for a long period of time.

The following analyses of three geyser-waters, selected from the Upper, Lower, and Norris geyser basins, may serve to show the composition of all of them, the differences which exist being equally well marked in the analyses of any two waters from the same geyser-basin.

	Bee-hive Geyser.		Fountain Geyser.		Fearless Geyser.	
	Grammes per kilo. of water.	Per cent. of total matter in solution.	Grammes per kilo. of water.	Per cent. of total matter in solution.	Grammes per kilo. of water.	Per cent. of total matter in solution.
Silica	0.3042	25.12	0.3315	23.69	0.4180	25.60
Sulphuric acid	0.0271	2.24	0.0195	1.39	0.0367	2.25
Carbonic acid	0.0920	7.60	0.2307	16.48	0.0046	0.28
Phosphoric acid			0.00004			
Boracic acid	0.0145	1.20	0.0138	0.99	0.0223	1.36
Arsenious acid	0.0011	0.09	0.0027	0.19	0.0022	0.14
Chlorine	0 3894	32.15	0.3337	23.84	0.6705	41.06
Bromine	Trace		0.0004	0.03	0.0026	0.16
Iodine						
Fluorine						
Hydr. sulph			Trace		Trace	
Oxygen (basic)	0.0364	3.00	0.0654	4.67	0.0113	0.70
Iron	Trace		0.0002	0.01	0.0006	0.04
Manganese			Trace			
Aluminium	0.0029	0.24	0.0057	0.41	0.0002	0.01
Calcium	0.0039	0.32	0.0014	0.10	0.0092	0.56
Magnesium	0.0002	0.02	0.0010	0.07	0.0001	0.01
Potassium	0.0213	1 76	0.0379	2.71	0.0415	2.54
Sodium	0.3118	25.74	0.3522	25.16	0.4046	24.77
Lithium	0.0061	0.50	0 0035	0.25	0.0081	0.50
Ammonium	0.00021	0.02	0.00015	0.01	0.00025	0.02
Cæsium					Trace	
Rubidium					Trace	
	1.21111	100.00	1.39979	100.00	1.63275	100.00

Bee-Hive Geyser, Upper Geyser Basin; date of collection, September 1, 1884; temperature, 199.4° Fahr.; reaction, alkaline; specific gravity, 1.0009.

Fountain Geyser, Lower Geyser Basin; date of collection, August 24, 1884; temperature, 179.6° Fahr.; reaction, alkaline; specific gravity, 1.0010.

Fearless Geyser, Norris Geyser Basin; date of collection, August 18, 1884; temperature, 190.4° Fahr.; reaction neutral; specific gravity, 1.0011.

The differences of temperature shown in these three waters are simply due to the varying interval between the time of collection and the last preceding eruption of the geyser. In the case of the Fountain, the water rises in a large open basin, which slowly fills up, increasing in temperature until the time of the eruption, the form of the basin permitting the collection of the water two or three hours before the next outburst. In the case of the Fearless the surface-reservoir is a shallow saucer-shaped basin, into which the water seldom rises before attaining a temperature near the boiling-point. At the Bee-Hive the water only reaches a sufficiently high level to permit of its collection without difficulty when the temperature stands at or near the boiling-point.

Dr. Raymond has made the suggestion that the addition of caustic

alkali would possibly precipitate some of the mineral ingredients found in these waters, thereby changing their chemical composition sufficiently to affect the point of ebullition. At the same time he remarks that the geyser-waters are probably too dilute solutions to be much influenced by such additions. Any one who glances at the analyses of the waters of the Bee-Hive, Fountain, and Fearless must see, I think, that they are not only too dilute to undergo any marked change of temperature, but that the mineral constituents consist mainly of the carbonates and chlorides of the alkalies, associated with a relatively large amount of free silica which would remain unacted upon by caustic alkali. There is nothing in the waters to be thrown down by the addition of alkali or permit any chemical combinations to be formed by the addition of a small amount of soap. The desire of tourists to "soap a geyser" during their trip through the Park grows annually with the increase of travel, so much so that there is a steady demand for the toilet-soap of the hotels. If visitors could have their way, the beautiful blue springs and basins of the geysers would be "in the suds" constantly throughout the season. Throwing anything into the hot springs is now prohibited by the government authorities. It is certainly detrimental to the preservation of the geysers, and the practice cannot be too strongly condemned by all interested in the National Reservation.

THE PRESENT STATUS OF ELECTRIC TRANSMISSION OF POWER.

BY RICHARD P. ROTHWELL, NEW YORK CITY.

(Buffalo Meeting, October, 1888.)

AT the Boston Meeting of the American Institute of Mining Engineers in February last, Mr. George W. Mansfield read an interesting paper on "The Electric Motor in Mining Operations," and he entered into the calculations of power required and cost of transmitting energy in great or small quantities to a considerable distance. I do not intend to go over this ground, or to enter into any discussion of the technical details of transmitting power by electricity, but merely to set up a milestone on the road of progress to show where we now are, how far we have come, and if possible

endeavor to make out the way for a short distance in advance of us. I shall merely allude to what has been done and what is being done in a few cases of interest to the mining profession.

The first case with which I am acquainted of the electric transmission of energy for mining purposes was at a stamp-mill in New Zealand, and was described in the *Engineering and Mining Journal* and other papers a few years ago. Since then this means of transmitting power has been much discussed, and some practical applications of it have been made. The fact of the transmission of vast amounts of energy to great distances over a slender thread of metal is still a marvel, interesting to every observer, expert or non-expert. It seems but yesterday that the first long-distances transmission from Creil to Paris was made as an experiment.

The vast extension of the system was at once suggested, but so long as the handling of high potential currents was deemed very hazardous, the size of conductors using low potentials made the cost of the copper almost prohibitory for far transmission.

Mr. Mansfield pointed out the well-known fact that the size or cost of conductors decreases as the square of the potential increases; but even so short a time ago as in February last he confined his calculations for the cost of transmitting energy to the use of 560 volt pressures, and "placed the limit at 4000 volts." Moreover the cost which Mr. Mansfield gave for a plant of given capacity was much higher than some of the manufacturers have since been contracting at and there can be little doubt that within a year even the present cost will be much reduced.

From information furnished me by some of our best manufacturers of electric plants, I am able to cite some general figures for the cost of an electric plant for the transmission of power up to ten miles. The figures are calculated on the basis of 100 horse-power units: the engines or turbines costing about $40 per horse-power; dynamos, $30 per horse-power; motors, $40 per horse-power, and using a current of 1000 volts.

For one mile distance an efficiency of about 80 per cent. will be guaranteed, and the cost per horse-power of the electric plant will be from $80 to $85, and the total cost per horse-power, including motive power, either steam-engine or turbine, will be from $120 to $130.

For transmission over five miles an efficiency of nearly 70 per per cent. will be guaranteed, and the cost of electric plant will be about $135 per horse-power, and the total cost, say, $190 per horse-

power. For a distance of 10 miles the efficiency of 50 per cent. to 55 per cent. will be guaranteed, and the cost of electric plant will amount to from $200 to $210 per horse-power, and the total cost, including turbine and line, say, from $280 to $290 per horse-power.

These approximate figures will give an idea of what responsible contractors will guarantee now, and thus show the present status of the art. Each individual case has its own conditions which may vary these figures somewhat. The weight of the uncovered wire per horse-power, transmitted with an electromotive force of 1200 volts at the terminals of motor, would be about 50 pounds for 10,000 feet per horse-power.

The potentials used in the following calculations are only 1000 volts. It is perfectly possible to go above these, and thus economize copper. Or the efficiency figure may be lessened, in case the primal energy is abundant, thus again saving copper up to the point where increase in cost of dynamo plant equals the saving in copper. Incidental expenses, difficult to generalize, such as freights, etc., are to be added to any estimate, covering material only. Should the present high price of copper be diminished, the cost of such installations would be considerably lessened. On the other hand, if conditions of excessive dampness exist, it may be necessary to add a considerable sum for insulation.

Take, for example, a 10-mile transmission of 100 horse-power. The cost of the line wire is put down by Mr. Mansfield at $200,000, while in the figures of eminent contractors it is given at $11,000, or but little over one-twentieth of Mr. Mansfield's estimate.

It is evident that if the voltage can be increased from 1000 to 10,000 or 20,000 volts, the cost of the line wires, which form the heaviest item in a long transmission, will be so greatly reduced as to make this method of transmitting energy far less expensive than those now in general use. And the influence of this upon mining, metallurgy, and on general engineering practice will be incalculable. Thus far the electric transmission of power has been made over only moderate or very short distances.

The first experimental plant transmitted 40 horse-power from Creil to Paris, 70 miles, with a loss of about 44 per cent.; but in actual engineering practice the transmission has been confined for the most part to distances much less than ten miles.

The New Zealand application of electricity in mining, to which I have referred, consisted in driving a stamp-mill by the power of a

waterfall distant some miles from the mill. In this country the longest circuit for mining work is that on the Big Bend of the Feather River in California, where the Sprague Company has put up a plant of motors for operating pumps, etc., on a circuit of about 18 miles. The power is obtained from turbines placed at the end of the tunnel that cuts off the 20-mile bend in the river.

This plant is reported to be working very successfully, though as the season during which it can work is very brief, it requires, and we believe possesses, extremely rich gravel to pay for pumping and sluicing. At Aspen, Colorado, there has been for some time past in successful operation an electric plant driven by a neighboring water-power, whose energy is thus utilized to pump and hoist underground in the mine and to haul on the main entry. The plant is very successful, and is said to be much more economical than steam-power, even where coal is cheap.

The latest installation in mining work is the very interesting one now being put in by the Brush Electric Company in the Comstock Mines, Nevada, which has been described in the *Engineering and Mining Journal*, of June 16th and September 22d, 1888.

The plant is to be used to drive the new Nevada mill, hoisting engines, etc., and it consists of six Pelton impact water-wheels, 40 inches diameter each, connected directly on to the shaft of a dynamo, of 135 horse-power, and these drive six electric motors of 90 horse-power each.

The dynamos and water-wheels are placed in a chamber excavated in the mine at the level of the Sutro Tunnel. The water is brought down the shaft from the surface and goes through a half-inch nozzle on to the water-wheel under the enormous head of 1630 feet, the heaviest pressure I believe, ever put upon a wheel. The power is brought back to the surface over a copper strand about three-eighths inch diameter.

The dynamos are of the constant current type, wound for 40 ampères. They are said to be absolutely automatic in the government of their current strength, no current regulator of any sort being required. The variation of current between full load and no load is less than two per cent.

The efficiency of both machines, the dynamo and electric motor, is considerably over 90 per cent.

It is not necessary here to repeat the descriptions of the various electric plants that have been installed in this country. Besides those already mentioned, we have a mill in Idaho, driven by elec-

tric motors. In Alaska it is proposed to bring power to the Treadwell 240 stamp-mill on Douglas Island from a great waterfall on the main land some miles away. We have electrically driven underground locomotives, hoisting-engines, coal-cutters, rotary rock-drills, and electric percussion-drills, which are said to be a proven success, also. Electric lighting of mines and mills is becoming common, and nearly every town has, or proposes to have, electric tram-cars in its streets, and we have also electric ore-separators for concentrating magnetic iron ore. In addition to electric lights, small motors in great numbers are being introduced to perform a variety of work in our houses.

We have many systems of "electric amalgamation," in which the rôle of electricity appears to be chiefly for the "cleaning" of the mercury by preventing its coating with an oxide film.

The electric smelting in the Cowles' aluminum process has long been a great success, and is about to receive an enormous development.

Electric welding, at first confined to small pieces of metal, is now effective and economical with quite large plates and bars, and with various metals.

It would be impossible in the space permitted me to enumerate all the applications which electric transmission of energy has received of late. The progress made in this department of engineering within the past year or two has been simply marvellous, and it requires no prophet to foresee that the same rate of progress continued a few years more will create a veritable revolution in engineering practice.

The questions to be settled are whether the electric transmission is more economical than the other means now in use, and finally, what is the prospect of reducing the cost in either of the systems.

The rapid adoption of electric transmission is a satisfactory demonstration that it offers substantial advantages over other systems, but there is no other system adapted to far transmission. The practical limit for wire rope transmission is quickly reached, and that by means of hot water under high pressure, as now being very successfully introduced in Boston, is far more expensive in first cost than an electric plant. Mr. A. V. Abbott's paper (*Trans.* xvi., 870) describes fully this Boston hot-water distribution.

The only far distribution of the large amounts of power required in general mining operations, by means of compressed air, with which I am acquainted, is that at the Chapin Mine, Michigan. The use of compressed air for driving-drills, etc., in long tunnels in this

country and Europe, required comparatively small amounts of power. The Chapin mine example is, without doubt, the largest plant in the world for the transmission of power by compressed air.

The plant in question furnishes power to the Chapin and Ludington Mines at Iron Mountain, Michigan. At the Quinnesec Falls of the Menominee River, about three miles from the mines, there are four turbines, one of 1000 and three of 900 horse-power each. Each turbine runs a pair of compressors. The air is compressed to about 60 pounds at 60 degrees Fahrenheit, and is carried to the mines in a 24-inch iron pipe, made in lengths of 68 feet, with expansion-joints every 10 lengths. This pipe cost $60,000 complete, each expansion-joint costing $500. The power is applied at the mines by supplying the air to Corliss engines for running Cornish pumps, hoists, etc, and is also applied direct to the rock-drills and to the pumps and to Corliss engine running tramway cable underground. The total cost of plant is estimated at about $500,000. A test made February 20th, 1888, shows the following results:

Indicated horse-power of compressors, ...	1,430.27
Sum of indicated horse-power of engines at the mines, .	390.17

Therefore only 27 per cent. of the horse-power indicated by the compressors is recovered as indicated horse-power in the cylinders at the mines:

Cubic feet of air at compressors in eleven hours, . .	1,231,736
Cubic feet at engines, ...	967,391
Loss in cubic feet, ...	264,345

The loss of pressure between compressor and engines is about two pounds.

In these figures we see that the loss is nearly 73 per cent. This includes the losses due to the leakage, and loss of energy in the form of heat, but does not include friction of engines or compressors; so that the total loss, between the point of application of power of the turbines, and the point of application of power of engine to shafts, etc., cannot be less than 75 per cent. of the total. In other words, the efficiency of the system stated, as would be stated for an electric system, does not exceed 25 per cent. The figures quoted above are a fair example of tests which are made at regular intervals.

The two mines are charged each with its share of the power used. The amount used in each mine is found by indicating all its engines,

and the expense is pro-rated from these results. The mining companies consider that the plant supplies them power at about the same expense as if they used fuel, which at this point is worth about the same as coal at $5 a ton.

The following figures may also be of scientific interest: The air leaves the compressor at a temperature of 330 degrees Fahrenheit, and at the exhaust of the engines and pumps ice is formed which would close up the exhaust opening in the pump were it not for a coal fire which is kept burning beside it. By comparing the compression curves of compressor cards and the expansion curves of the engine cards with the curves of isothermal compression and expansion, there appears a loss in compressors of over twenty per cent., and in the engines of over twelve per cent. These are evidently losses which no possible device in air compressors can materially decrease.

In these mines, as in some other cases, the use of compressed air does not have any great advantage as regards ventilation, as there is a natural circulation of air, which is usually sufficient, and it is artificially increased whenever desired by the use of current produced by running water.

From the data given above it is easy to institute a comparison between compressed air and electric transmission in this case, but such comparison should not be applied too generally, for there are conditions that greatly vary the efficiency of each system, and these conditions should be weighed in each case. There can be no doubt, however, that with a pressure of even 1000 volts, electric transmission is, for long distances and under ordinary conditions, much the cheaper method of the two.

If it be possible to transmit currents of high voltage, say 10,000 or 20,000 volts, as is now intimated has been done in London by the Ferranti system of insulation, then the problem assumes an entirely different phase, and the field for the application of electric transmission of power is widened to an incalculable extent.

I am not in position to affirm or deny the possibility of transmitting with safety currents of such enormous pressures, but the progress in the electric art has been so exceedingly rapid during the past few years that we may be permitted to believe that means for controlling these high voltages will be devised, and that they will be found in practical use before many years.

On the assumption that 10,000 and 20,000 volt currents can be handled with safety and economy, every waterfall in the country,

even in its inaccessible parts, would become of value and would contribute to the efficiency of our people.

The waste gases from our 35,000 coke-ovens could be used to generate and drive dynamos, until such time as a practical method of converting the heat of combustion directly into electricity shall have been perfected.

Perhaps the most momentous vision which is opened to our view by this anticipated transmission of high-pressure currents is in the influence it may have upon our coal and railroad industries.

There is no branch of engineering which is so backward, so disgraceful to the engineering profession, as that which relates to the utilization of the heat units contained in fuel. Almost the only application which attains a satisfactory economy is the iron blast furnace. The percentage of the heat units evolved in the complete combustion of coal that is actually utilized varies from perhaps 3 or 4 per cent. in many open grate fires to perhaps 12 or 15 per cent. in steam boilers. It may be roughly assumed that the average result obtained in burning the 140,000,000 short tons of coal estimated to have been produced in the United States this year (1888) was not better than 8 pounds per horse-power per hour.

The value of this coal averaged about $1.40 at the mines, and the average actual cost to consumers throughout the country could not have been less than $4.20 a ton, or three times the cost at the mines; or, at the assumed efficiency of 8 pounds per horse-power per hour, say 1.7 cents per hour per horse-power, as compared with .57 per cent. at the mine.

If, instead of a consumption of 8 pounds per horse-power, this coal had been used with an efficiency of 2 pounds per horse-power, which is quite practicable, the cost would have been .4 cent per horse-power per hour at the point of consumption, and .13 cent at the mines. In other words the cost of the fuel per horse-power to the consumer averages probably ten times what it would at the mines, if using electricity generated by steam-power. This is roughly the margin which an electric transmission would have to work on, and since we have assumed the average cost of transporting a ton of coal from the mines and distributing it to consumers at $2.80, we have here a unit of measure.

Without entering at this time into detailed estimates of the cost of transmission of the vast amount of power represented by the combustion of millions of tons of coal, the high efficiency already attained in electrical transmission makes it not improbable that before very

many years have passed we shall see a great part of our coal burned at the mines in specially designed furnaces, and the energy there generated carried as electric currents of high voltage to the great centres of consumption, and then reduced and distributed to manufactories and to public and private buildings, where, reconverted into heat, it will take the place of coal in cooking, heating, lighting, motive power, etc., with many evident advantages in cleanliness, safety, convenience and economy.

The cost of transmitting the energy contained in the coal from the mines to the points of consumption will certainly be far less than the cost of carrying the coal on railroads and distributing it by carts; and the efficiency of the coal would be vastly greater when burned in specially designed furnaces than in the wasteful appliances in use now by the average consumer.

The electrical transmission would give as much useful effect from one ton as is now obtained from two or three tons, so that it would leave from $5.60 to $7.40 as the margin with which to work; and its adoption would prolong the duration of our supply of anthracite, which is being so rapidly exhausted by the wasteful methods of mining now in use.

Discussion.

Fred. G. Bulkley, Aspen, Colo. (Communication to the Secretary): The following details of the Aspen plant, referred to by Mr. Rothwell, may be of interest. Two Pelton water-wheels, to which water is delivered under a head of 68 feet through 4-inch nozzles, supply the power. Assuming an efficiency of 85 per cent., each wheel yields 27 horse-power. By means of belt-pulleys on the shafts of the Pelton wheels, the power is applied to a 500-volt, 80-ampere, Edison dynamo, equivalent to the conversion of about 53 horse-power. The distance to which the power is transmitted is 7300 feet. For the distance of 6000 feet, on the surface, a No. 2 copper-wire is used, while for that part of the line which is underground, 1300 feet, a cable was put in. It is $\frac{1}{8}$-inch in diameter, consists of seven No. 10 copper-wires, is waterproof, and thoroughly insulated, "bitite" being the insulating material. Under these conditions an efficiency of 70 per cent. is obtained at the two Sprague motors. The plant has been in constant use since May 5, 1888, and has proved very satisfactory in all respects.

I had been under the impression that the electrical transmission

of power was applied to mining work by the Compagnie de la Ferroniure in the Loire Valley, in France, about the year 1880. If that is the case the New Zealand mill is antedated.

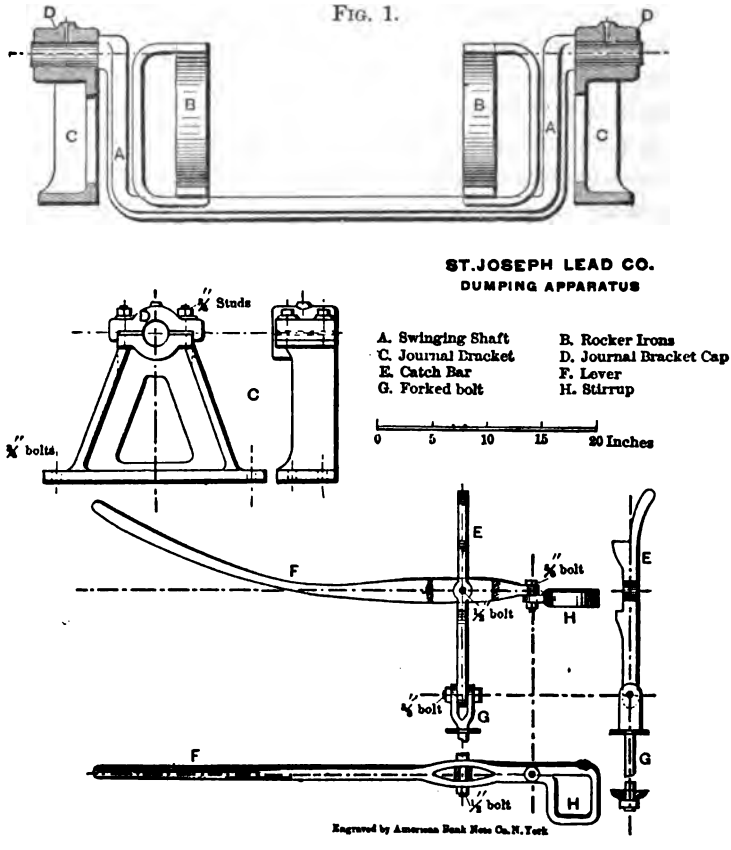
AUTOMATIC DUMPING-CRADLES FOR MINE-CARS.

BY H. S. MUNROE, SCHOOL OF MINES, NEW YORK CITY.

(New York Meeting, February, 1889.)

THE accompanying figures illustrate the dumping-device used at the new dressing works of the St. Joseph Lead Co., at Bonne Terre,

FIG. 1.



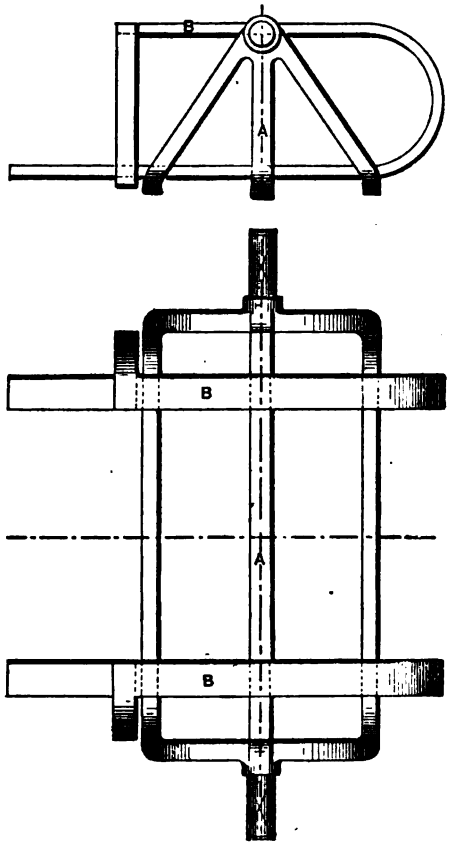
End-views and Section.

Mo. It is a dumping-cradle, of a type much used abroad, in which the car is turned upside down to discharge its load. The main

advantage of this type of dumping-apparatus, is that it permits the use of cars of simple construction, not weakened by hinged doors or bottoms.

The wheels of the mine-car run between the upper and lower arms of the curved rocker-irons *B*, the lower arms forming a con-

FIG. 2.



Dumping-cradle; Side- and Top-views.

tinuation of the track. The apparatus is so constructed that the axis *D* is below and back of the center of gravity of the loaded car, but above the center of gravity of the empty car. The dumping of the car, and its return to a normal position are both automatic. A chain, not shown in the sketch, prevents the cradle from turning too far, and the catch permits the car to be held upside down till the load is completely emptied.

The dimensions of the car for which this apparatus is designed are as follows:

Height over all,	3 ft.
Width of car-body,	2 ft. 9 in.
Length of car-body,	3 ft. 3 in.
Depth of car-body,	1 ft. 8 in.
Bottom of car-body above track,	1 ft. 4 in.
Diameter of wheels—tread,	11½ in.
Diameter of wheels—flange,	14½ in.
Distance between axles,	1 ft. 3 in.
Gauge of track,	2 ft.
Capacity of car,	1400 pounds of ore.

The body of the car is rectangular, and is made of $\frac{3}{16}''$ boiler-plate, the sides and ends in four separate pieces, bent at the corners and riveted. The top edge is strengthened by a 3″ piece of strap-iron $\frac{3}{8}''$ thick, fastened by rivets. The bottom is attached to the sides by a piece of angle-iron riveted through the flanges and placed inside the sides and ends and below the bottom-plate. The car-body is supported on two oak sills, 4″ x 8½″ by 22″ long. These raise the bottom of the car high enough to allow it to project over the wheels.

I have an indistinct recollection of seeing, at a mine near Georgetown, Colo., probably the Dunderberg, a dumping-cradle similar to the one above described.

Mr. Edward F. Weekes, a student of the graduating class of the School of Mines, Columbia College, has prepared at my request from notes taken during the work in 1888 of the summer class in Practical Mining, the following account of a dumping-cradle employed at the Chapin mine, Lake Superior.

The dumping-cradle used at the Chapin mines is an example of a type in common use abroad. Like the one used at the St. Joseph mill, it dumps the car by turning it upside down, avoiding the necessity of gates in the side or bottom of each mine-car, and thus permitting the cars to be made light, stiff and strong at reduced cost.

One disadvantage of dumping-cradles is that, as ordinarily constructed, they are fixtures. A separate cradle has to be erected at each dumping-place. This difficulty is overcome at the Chapin mine by the simple device of mounting the cradle on wheels.

The cars, loaded from ore-chutes in a foot-wall drift at the sixth level, are conveyed to the foot of the shaft, through a cross-cut

tunnel, by a slow-moving endless rope, to which they are attached singly, by very simple and effective grips. At the surface, the cars are run into the dumping-cage, and the ore is dumped into the bins from which the railroad cars are loaded.

The ore-cars, 6 feet 1 inch long, 3 feet 4 inches wide at the top, 2 feet 4 inches wide at the bottom, and 2 feet 4 inches deep, are made of ¼-inch plate-iron and have a ¼-inch strip riveted around the top, and also one around the sides and bottom at the middle. The wheels, of 8-inch radius, 3 feet gauge, are set 2 feet apart. There are two 3-inch x 3-inch iron bumpers, 4 inches long, at each end of the car. The capacity of the car is two tons.

The endless wire rope used for hauling the cars, ¾-inch in diameter, and moving about ¾ of a mile per hour, is carried between the rails on grooved rollers, 4 inches wide and 8 inches in diameter, and is guided around the curve from the cross-cut into the drift by seven grooved wheels, one side of the groove being 14 inches and the other 12 inches in diameter. The first of these wheels is set inclined, the second horizontal, the third inclined, the fourth horizontal, etc.

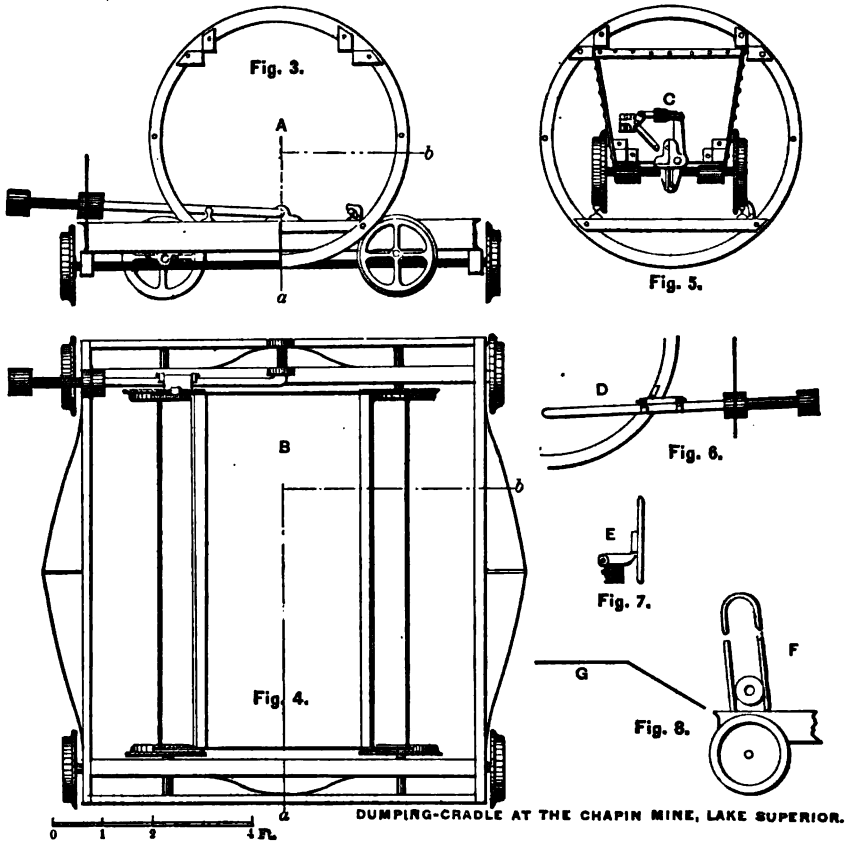
The motive power, located in an excavated room on the right of the cross-cut and near the shaft, consists of two 12-inch x 36-inch Corliss engines, run by compressed air. There are two sheaves or drums, one fixed to a cog-wheel which receives the motion, and the other set in a frame, held down by bolts working in slots, 15 inches x 1½ inches. A 3-inch jack-screw is set between the end of the frame nearest the first sheave and a cross-piece firmly bolted to its foundation. By this the distance between the two sheaves can be varied as the stretching of the rope may require. The rope passes around these two sheaves four times.

The cars are attached to the rope by a clutch or grip (Fig. 5), which extends 7 inches below the car-bottom. The rope is gripped between a piece bolted to the car and a lever (short arm 7½ inches, long arm 11½ inches), held in position by an elbow-joint.

The cars are held in place in the double shaft, on cages 6 feet x 7½ feet in size, by two clutches, which, when in place, rest upon the rails and between the wheels.

At 20 feet above the ground-level a trestle, about 120 feet long, leads from the shaft-house to an ore-pocket. From about 25 feet in front of the shaft to directly over the ore-pocket, extends the track for the automatic dumping-cradle, shown in Fig. 3, which is an end-view with the portion *a b* removed, and Fig. 4, which is a plan. The

frame (8 feet 4 inches wide and 9 feet 8 inches long, made of 7½-inch x 2-inch channel-beams with two 4-inch I-beams of the same depth set across the frame 7 inches from each end) rests on four wheels of 8-inch radius, running on a track of 8 feet 11 inch gauge. The cradle proper consists of two iron rings 5 feet 4 inches outside diameter, 3 inches wide, and ⅜-inch thick, held apart by two angle-irons, so set that the horizontal distance between them shall be the width



DUMPING-CRADLE AT THE CHAPIN MINE, LAKE SUPERIOR.

of the top of the car (see Fig. 5). The track-rails are carried on iron cross-pieces. The cradle, which is also strengthened by two iron rods diametrically opposite each other, rests between the I-beams, on four wheels of 9-inch radius, the axles of which are supported in bearings on the under side of the two end cross-beams. The distance between the centers of these wheels is such that the rails in the cradle correspond with the rails which project over and rest upon

the frame, when the cradle is in position to receive a car. The cradle is free to revolve on these wheels.

There is also a catch for the wheels of the car similar to that on the cage. Fastened to one end and revolving around the central axis of the frame, with the bearing shown in Fig. 4, is a square iron rod, which projects 18 inches beyond the side of the frame. On the end of this rod is a wheel 5 inches wide; and over the side channel-bar is another wheel of the same size. The part of the rod between these two wheels is round. At a point on this rod opposite the end-ring is a hinged piece which projects under a catch on the cradle-ring (see D and E, Figs. 6 and 7); the inner wheel is confined in an inverted U-shaped guide, fastened on the side channel and beam set inclined; a plank running lengthwise, on the other side of the frame, acts as a platform. An endless wire rope, carried from each end of the frame over a system of pulleys to the engine-house, more than 750 feet away, moves the apparatus. The 12-inch x 36-inch Corliss engine, driven by compressed air, runs the truck at shaft B and the cradle at shaft C.

In the cross-cut underground, at some distance from the shaft, the tread of the moving car-wheels moves a piece of angle-iron, hinged directly alongside of the rail, and, when in place, having its free end 5 inches above the top of the rail. The motion of this angle-iron, by means of a wire and bell-crank, rings a bell at the shaft, announcing the approach of the car in time to ensure its proper detachment from the rope. The car is pushed by hand a few feet to the cage, and sent to the surface, where it is pushed off the cage and upon a truck 5 feet long, running at right angles to the track, and is then shifted to a track midway between the two cages. From this track it is pushed upon the cradle (the track projecting over and just clearing the cross-beams of the cradle-frame, and meeting the rails in the cradle). The cradle is now moved on its track until directly over the ore-pocket, at which point the wheel on the end of the rod projecting beyond the side of the cradle, strikes an incline G, Fig. 8 (the incline is not drawn to scale), which gives it a sudden upward movement and starts the cradle revolving. The center of gravity of the loaded car and cradle being above the axis of the cradle insures the completion of at least a half-revolution, which would dump the ore; but the momentum carries the cradle beyond this point. A workman, who always rides out on the platform of the frame, now rights the cradle, and it immediately returns to the

shaft. The incline mentioned above is made of wood covered with a piece of inch half-round iron, which acts as a track.

It is unnecessary to describe the manner of returning the empty cars into the mine, since it can easily be inferred from the foregoing description, the process being practically reversed.

In the platform back of the shaft-house is a second cradle, into which the cars are run in order to be turned bottom-side up for oiling.

No hoisting is done at night, the cars being loaded and stored on the track under the chutes until morning, when they are sent to the surface.

The ore from the working places is trammed by hand to the ore-chutes, which are located at the end of each cross-cut into the ore-body. The cars used for this purpose are $3\frac{1}{2}$ feet long, 2 feet wide at the top, $1\frac{1}{2}$ feet wide at the bottom, $1\frac{1}{2}$ feet deep, and open at one end. The wheels are of 6-inch radius, axles 15 inches apart, and the forward axle 18 inches from the front of the car. Each car holds about $1\frac{1}{2}$ tons and dumps over the front axle.

The system of handling ore above described is followed during the shipping season only; at other times, one-ton cars are used in place of the two-ton cars, and the ore is dumped from a trestle, running between the two shafts, upon an immense ore-wharf, about 1000 feet long and about 150 feet broad (estimated), from which it is loaded into the railroad-cars when shipments begin.

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*ORE-DEPOSITS OF THE BLACK HILLS OF DAKOTA.**

BY FRANKLIN R. CARPENTER, RAPID CITY, DAKOTA.

(New York Meeting, February, 1889.)

In area, the Black Hills are about equal to the State of Connecticut. As the accompanying geological map indicates, they exhibit in the main a simple structure, presenting a central mass of granite and Archæan rocks, on all sides of which the later sedimentary rocks dip

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* Note by the Secretary.—The substance of this paper, with considerable additions, constitutes a chapter in the report of Dr. Carpenter to the Trustees of the Dakota School of Mines, of which he is Dean. By the permission of the Trustees, it was communicated in advance to the Institute.

away, lying, in the main, conformably upon one another, and exposing, as the result of erosion, the successive edges of the Cambrian, Carboniferous, Triassic, Jurassic, Cretaceous and Tertiary, the Cretaceous constituting a nearly level plain through which the earlier strata have been uplifted. Of the Cambrian, no rocks occur except the Potsdam; and the Silurian and Devonian are entirely wanting —a remarkable fact, when considered in connection with the perfect conformability between the Potsdam and the Carboniferous. There is also a break between the Cretaceous and the Miocene—a period of great erosion.

Large veins of granite occur in the Archæan rocks of the southern Hills; and sheets of intrusive igneous rock, sometimes typical laccolites, are found crossing or intercalated in the sedimentary strata further north.

In a paper on the geology of the Black Hills, contained in my report to the Trustees of the Dakota School of Mines, I have discussed this subject in detail. The foregoing summary statement must suffice in this place, to introduce my description of the ore-deposits.

The metalliferous deposits of the Hills are both extensive and varied. Iron, copper and tin abound, as well as gold and silver. Cement, fire-clay and gypsum, as well as a great variety of building stones, exist in inexhaustible quantities. It is not so much the object of this paper to set these forth in detail, as to give such results of our study concerning their occurrence, nature and treatment as have been obtained at the School of Mines.

The great deposits of the Black Hills are of Archæan age, and yield mainly an auriferous pyrite, though lead, zinc and other ores occasionally occur. These deposits are sometimes lenticular in shape, and seemingly form independent members of the slate and schist series, and share in all their folds and contortions, having, like them, a columnar cleavage coincident with the bedding.

I. Placers.

Two kinds of placers, tin and gold, are found in the Black Hills. The placers are also of different ages, Quaternary and Recent, and that ancient consolidated placer found at the base of the Potsdam, for it seems impossible to call it anything else. The Quaternary placers yet yield some gold, but their richer parts, like Deadwood gulch, are practically exhausted. Upon some streams, where the

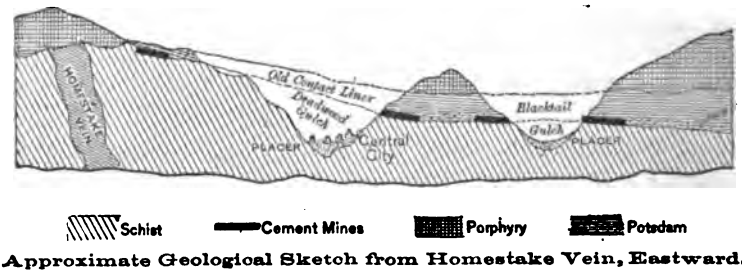
bed-rock is deep, requiring expensive pumps and other machinery, there may yet be found gold in quantity.

In the formation of the Potsdam rocks, the ocean advanced upon the slowly sinking Archæan island, sorting over the material worn from the land, rearranging and depositing it as sand and gravel, thus forming the basal member of these rocks. Containing particles of gold derived from the Archæan veins, it became in reality a gold placer, especially in the neighborhood of the great Archæan gold deposits, such as the Homestake. In time it was consolidated to rock, and to-day forms the "cement" beds of the miner. It is mined, stamped and amalgamated as other free-milling gold ore.

The sketch given in Fig. 1 was made by Mr. W. B. Devereaux to illustrate his paper on the occurrence of gold in the Black Hills.*

The former contact-line of the Archæan and Potsdam rocks, as well as the subsequent erosion, is clearly shown. The porphyry capping of the vein was injected before the Potsdam and overlying

FIG. 1.



Approximate Geological Sketch from Homestake Vein, Eastward.

rocks had suffered erosion. It flowed sometimes beneath the Potsdam and sometimes upon it and beneath the Carboniferous rocks.

The large body of porphyry upon the left is, at no great distance, still overlain by the Potsdam, which it lifted upward after the manner of a laccolite, while upon the east, west and south of the locality here shown, similar sheets of igneous rock disappear beneath the yet unremoved Potsdam and Carboniferous. There is every reason to suppose that the porphyry was injected while the Mesozoic rocks, even, were yet in position, although Mr. S. F. Emmons was led to say that, if the above cut is correct, the porphyry must be subsequent to the date of the erosion of the Cretaceous and Palæozoic rocks, and, therefore, probably post-Cretaceous. An examination of the district, however, leads one to the view above given.

The drawing represents a section taken perpendicularly to the

* *Trans.*, x., 465.

Homestake, and extends about one mile and a half to the east. The lower contact-line is the ancient Potsdam beach. The later placers occupying the gulches beneath derived their gold mainly from the disintegration of the overlying Potsdam; and while Deadwood gulch has received additional gold directly from the Homestake deposit, Blacktail gulch, lying, as it does, with Deadwood creek between it and the Homestake, contains no gold that came directly from this deposit, what it has having come through the medium of the Potsdam.

As Mr. Emmons remarks, the most interesting fact connected with this deposit is that it furnishes absolute proof of the Archæan age of the Black Hills gold. No doubt, the Homestake bed was then gold-bearing; but whether it has been subsequently enriched by the numerous porphyry dikes in it and parallel to it, I will consider presently.

Mr. Devereaux, who had the management of some of these cement-mines, observes that the gold in these conglomerates is finer than the gold of the Homestake, whence it was derived. Since it was set free, it has been subjected to various chemical agencies, which have dissolved out a part of its silver alloy. This solution of silver, having been confined to the surfaces of the pieces of placer gold, has given them a corrugated appearance, and this being true, it follows that the smaller the pieces are, the finer the gold. The average of Mr. Devereaux's assays gives the fineness of the gold as .904; of the silver, .096.

The different mines upon the Homestake, according to Mr. George Hewitt, yield gold in fineness as follows:

	Homestake.	Highland.	Terra.	Deadwood.	De Smet.
Gold.....	.820	.830	.825	.850	.820
Silver.....	.170	.155	.160	.140	.170

He also observed that, in the vicinity of the dikes which cut the Potsdam, but little gold was found; and such as there was, had lost its rounded appearance and gave every evidence of having been subjected to a powerful solvent. As this fact bears upon the formation of other gold-deposits found in these rocks, I call special attention to it.

II. Ore-Beds.

The Homestake and Similar Deposits.—These are the most extensive and important of the mineral deposits of the Black Hills. The

best known group of mines, occupying a small section 6000 feet long by 2000 feet broad, known as the "Belt," produced last year $2,271,341.14. That these deposits were originally laid down as beds, I have no doubt; but it is also certain that they were subsequently much altered. My view of their formation is as follows:

At the time when the enclosing rocks were laid down, large quantities of proto-sulphide of iron were formed through the agency of decaying organic matter, the presence of which at that time is attested by the large quantities of carbonaceous matter yet present in these beds. This proto-sulphide of iron was not (at least, not to any great extent) gold-bearing. Subsequently, by the action of ferric salts in solution, this proto-sulphide was changed to bi-sulphide, and the iron solutions which wrought this change also brought the gold which is now found in these deposits. I have adopted this view for the following reasons: The presence of such quantities of graphite is considered proof of former organic matter. These beds of pyrite often pass into pyrrhotite, which may be classed as a proto-sulphide of iron, and in so doing they always cease to be gold-bearing. I suppose the pyrrhotite to be nearer the original condition of the sulphide, and the change that has taken place may be represented by the following chemical equation:

$$2\,FeS + Fe_2(SO_4)_3 = FeS_2 + 3\,FeSO_4.$$

The part of the "Belt" belonging to the Homestake combination i gold-bearing for a distance of 6000 feet. The "ore" is not continuous throughout this distance, but occurs in shoots or vast "pipes," lenticular in cross-section. The beds of argillites, phyllites and amphibole schists, in which these shoots occur, strike north 37¾° west, which is also accurately the strike of the plane or "ore-channel" in which the shoots occur. The dip of the beds, as a whole, is to the east. The shoots dip east also, but athwart this plane, at an angle of about 45 degrees. The ore and enclosing rocks have indifferently the same general cleavage-structure.

In the Homestake proper, or south end of the Belt, there are numerous sheets of porphyry, or more properly, of felsite, sometimes cutting across the stratification, but usually parallel to it. In the northern half—the Deadwood-Terra and De Smet end—no porphyry is found in the vein; but the section was once overlain by felsite which yet remains as a capping to the ridge between Gold run and Bobtail gulch, and between Bobtail gulch and Deadwood creek. This porphyry was injected between the slates and the Potsdam, raising the latter and other overlying rocks after the manner of a laccolite,

though they have since, in the immediate area, been removed. The vent through which this sheet was injected had probably no connection with the small dikes found in the Homestake, but was intruded from the southwest, and most likely was connected with the igneous mass forming Terry's peak.

The percentage of pyrite impregnating the slates and schists and forming the "ore" is never large. The yield upon concentration is seldom 20 per cent. The average is probably not more than 7 per cent. There are no solid bodies of pyrite.

The influence of the porphyry upon the lode seems to be good, but whether it produced an enrichment of the bed, or simply rendered it, by some action, oxydizing or otherwise, more free-milling, I have not determined definitely. The fact, however, is that the Homestake yield per ton is much greater than that returned by the Deadwood-Terra and other claims where the igneous intrusions do not occur—the former, from the best attainable data, being $3.87 per ton, while the latter, as figured from official reports, lies between $2.03 and $2.82. The sheets of igneous rock in the Homestake are porous and much decomposed. Careful assays showed no gold. The sheets east of these, near the pump-shaft, are pyritiferous, and assay from $1 to $2 per ton. I am led to the conclusion that the porphyry has had a two-fold benefit:

First—It has rendered the ores more free-milling.

Second—It has produced in its neighborhood either an enrichment of the deposit or a further concentration of what gold originally existed in it.

That it was gold-bearing before the injection of these dikes is proved from the fact that, remote from their influence, it is gold-bearing, and that all similar beds, in other parts of the Hills where no igneous rocks occur, are also gold-bearing; as well as from evidence furnished by study of the Potsdam conglomerate, laid down prior to their injection. At the surface the ore has always appeared in shoots, as I have described; but I am credibly informed that in the deep workings of the Homestake end this is no longer the case, the bed being now continuously ore-bearing for many hundreds of feet. These ore-bodies are said sometimes to have exceeded 300 feet in thickness. Careful measurements of the big open cut upon the Golden Star claim gave as the cross-section of that body 150 x 350 feet, while the Terra cut gave 150 x 250 feet. These figures are probably correct, so far as the width is concerned; but the shoots may be longer.

This deposit may be compared to the Rammelsberg; but the

descriptions of Cotta, Groddeck and Wimmer show many differences. The enclosing rocks of the Homestake are Archæan, while the Rammelsberg deposit is found in the Devonian. The shape of the ore-bodies and character of the accompanying minerals also differ; and this deposit is longer and thicker than the Rammelsberg. Beginning with the northern end, it gradually widens to 350 feet, by actual measurement, at the Segregated Old Abe and Homestake. Other ore-bodies are found in the same beds, but occupying different planes. From the shaft on the Golden Terra extension, the Caledonia discovery-shaft lies east, 1300 feet, at right-angles to the course of the Homestake. Other ore-bodies also occur, but want of time has prevented my giving them special study. The Caledonia ore-body is a pyritiferous chloritic schist between a hanging-wall of phyllite and a foot-wall of mica-schist, in which occur layers of chloritic schist, also carrying gold, as the main body does. The tunnel intersects, before reaching the vein, several small dikes of felsite, the strike of which is the same as that of the ore-deposit. Two ore-bodies are worked by the company, one about 40 feet, and the other 80 feet, thick.

I consider the Belt, as a whole, to be a particular zone of slates and schists, in which there are many lenses and shoots of ore. These shoots are never solid bodies of pyrite, as in the Rammelsberg, but zones impregnated with pyrite. The whole thickness of the beds in which these ore-bodies occur is about 2000 feet. They can be traced, I believe, throughout the entire length of the Archæan rocks in the Hills, and are gold-bearing in many other places south of the Belt. East and west of it, the slates and schists forming the Belt pass into quartzites, which may be regarded as the hanging- and foot-walls proper of the deposit. A line drawn upon a given level through the ore-channel or shoots constituting the Homestake, Deadwood-Terra and De Smet ore-bodies is absolutely straight, showing that though these shoots vary in size, they all lie in the same plane.

Different methods of mining are pursued upon the Belt. The surface-ores are worked by large open cuts, and at a minimum cost; but as the depth which can be reached by this method, even where the deposit is from 150 to 200 feet wide, is limited, extensive underground workings exist upon all the mines except the De Smet. In the Homestake mines the ground, as fast as broken, is securely timbered by the Nevada "square set" and filled solidly with waste rock. At the Caledonia, no timber is used, the ground being safely supported by solid pillars and worked in a way which is a little different from that ordinarily pursued where timbering is avoided.

The method is original with the able superintendent, General Skinner, and can only be made plain by drawings. Ample opportunity was afforded me for study; but the results are reserved for a future paper upon the methods of mining in the Black Hills.

Cost of Mining at the Homestake.—For the year ending June 1st, 1888, the cost of mining at the Homestake, including the "mine" and different shafts, as figured from the annual report, was as follows:

Items.	Amount expended.	Cost per ton.
Labor	$265,266.51	$1.0900
Dead work	55,267.50	0.2271
Supplies	13,413.08	0.0551
Powder	2,685.05	0.0110
Candles	4,123.75	0.0169
Machinery	10,641.67	0.0437
Oil	3,098.90	0.0127
Timber	42,010.09	0.1726
Wood	12,267.00	0.0504
Coal	997.80	0.0041
Cost of mining	$409,771.35	$1.6836
Cost of milling*	202,951.27	0.8349
Total cost of mining and milling	$612,722.62	$2.5185

Leaving out the dead-work and the work done at the various shafts, and including only the disbursements at the mine for labor, supplies, etc., there was expended $336,596.31 which gives as the cost per ton of ore mined, $1.38.

There were mined 243,355 tons of ore, which yielded in gold, $3.68, and in silver, $0.03; total, $3.71 per ton. The net profit per ton, including all expenditures was $1.19.

The cost of mining at the Highland and Deadwood-Terra, judging from the average yield of the ore, must have been very much less than this. The difference is probably to be accounted for by their greater amount of open-cut work.

Other Deposits.—Aside from the Homestake, other deposits of this class have been but little worked. Numerous beds of similar pyritiferous gold-bearing schists exist throughout the Hills, which, so far as surface-indications go, cannot be distinguished from parts of the Homestake belt. In early days, numerous small stamp-mills were built; but from various causes they failed to pay. If the Homestake were worked in a small way, it also would not pay. It

* See Prof. Hofman's paper on "Gold-Milling in the Black Hills," p. 498 of the present volume.

is the enormous scale upon which these mines are worked, thus reducing everything to a minimum cost, that enables them to run at a profit.

That deposits carrying free gold to nearly or quite the same extent as some of the Belt mines exist in other parts of the Hills, is well known. Many samples, not "specimens," were taken during the progress of this work, which, at the School of Mines, returned by free-milling tests from $2 to $12 per ton, this last amount having been returned from claims in both Custer and Pennington counties. As free-milling ores can be tested by any miner with a common mortar and pan, we did not devote the same time to their examination as to those ores which require more technical knowledge for their treatment. These last deposits, save in the gossan or outcrop, show little or no free gold. Beyond this, they do not differ from the deposits already described, and, like them, carry ores in immense quantities. If they can be successfully treated, the gold-output of the Hills will be materially increased.

I have spent much time in the study of these refractory ores. If the gold was in the sulphuret, I saw no reason why it might not be concentrated to a valuable product. If these bodies were solid pyrite, of course no concentration could be made. If in small veins, the cost of mining would be too great, for their value seldom reaches $6 per ton. It seemed that a hint might be taken from the large mines of the Lake Superior region, where, owing to the great quantity in which low-grade copper-ores occur, as well as to the ease with which they can be concentrated, enormous dividends have been paid from ores worth far less, as they are broken from the mine, than those of which I speak.*

* The Atlantic copper-mine of that section made in 1885, 1886 and 1887, the following extraordinary showing:

The rock yielded only 0.743 per cent. of refined copper, and was treated at the following cost per ton:

	1885.	1886.	1887.
	Cents.	Cents.	Cents.
Mining, selecting, breaking and all surface expenses	78.62	80.88	87.23
Transportation to mill	4.80	3.48	3.80
Stamping and separating	30.36	26.53	27.31
Freight, smelting, marketing and New York expenses	25.45	24.25	23.07
Total working expenses	139.23	135.14	141.41
Total expenditures	143.30	138.01	145.22
Net profit	22.05	15.29	30.53

Of the vast quantity of these pyrite ores there was no doubt, for the beds seldom showed less than 50 feet in thickness and could be traced for thousands of feet. Could rock yielding less than $5 in gold, not free-milling, be made to pay? In the study of the question, the results obtained were not always satisfactory. While we have in the metallurgical laboratory ample machinery, I could not treat any of the lots upon a working-scale; but I found some deposits which might be made to give profitable employment to hundreds of men. Not wishing to appear to be advertising particular claims, I give a few results, only omitting names.

One deposit, belonging to a large group in Pennington county, is more than 100 feet wide, and can be clearly traced for 4500 feet. The ore appears to be a hornblendic schist, which in places gives place to a mica-schist, and this often to a highly plumbaginous schist, the character of which has not been definitely determined. These are impregnated with ordinary pyrite and arsenopyrite. The creek, for some distance below the crossing of the bed, has been extensively worked for placer-gold, which I believe to have been derived from this deposit. Many tests were made from different parts of the ore-bed, with good results. One sample from the main workings, average value of rock, $6.19, yielded 22.15 per cent. of concentrates, assaying $22.73, of which $20.46 was saved by chlorination. That is, 4.51 tons of rock yielded one ton of concentrates, returning $20.46. The cost of mining, transportation to mill, taxes, etc., for 4.51 tons of rock at 78.62 cents per ton (same as at "Atlantic" copper mines) would be $3.545; of concentration for 4.51 tons at 30.36 cents, $1.369; of chlorination per ton of concentrates, $7; incidentals, exchange, etc., $1; total cost per ton of concentrates, $12.914; leaving as total profit on one ton of concentrates, $7.546; or on one ton of rock, $1.67.

In another instance, mica-schist, carrying pyrite, formed a zone 50 feet wide, adjoining a granitic tin-vein. The rock yielded 25 per cent. concentrates, worth $43.40 per ton.

The experiments made upon Custer county gold-ores were frequently very satisfactory. In one instance of exceptional richness (mica-schist with pyrite), the value of rock per ton was $24.84; per cent. of concentrates, 9.41; value per ton of concentrates, $142.56. A sample from another similar deposit gave ore assaying $12 and concentrates worth $67 per ton. A third yielded 25 per cent. of concentrates, worth per ton $42. A fourth (black micaceous schist, with arsenopyrite in a zone 150 feet wide), yielded rock,

$3.10 per ton; per cent. of concentrates, 9.40; value of concentrates, $17.56 per ton. This should have been, with good concentration, very much greater. I am unable to account for the loss; but, notwithstanding this, the size of the deposit, the facility for working, etc., would yet leave a profit. The belt probably extends through the county, if not the Hills, and is believed to be a continuation of the zone from which the first Pennington county sample above was taken.

Prof. Hofman's paper, already referred to, gives particulars concerning the Homestake concentrates.

A well-known mine, now not worked, was investigated with the following result: large quantities of ore are exposed in this mine, which has been worked intermittently for free gold for the past seven or eight years. After the free gold was extracted, I found the ore yielded 10 per cent. of concentrates, which averaged $17 in gold per ton. The free gold formerly saved paid for all development and running expenses. The amount received from the concentrates, after deducting the cost of chlorination, should be clear profit. From a careful survey and examination, I estimated that the opening of another level would add $300,000 to the amount of ore in sight, one-third of which should be clear profit.

It is the intention of the School of Mines management to devote the coming season to the further survey and study of the deposits of this class. I am satisfied, from the work of the past seasons, that these ores will pay to work in the manner indicated, viz., by concentration and chlorination, and upon a scale at present little imagined. The enormous quantities in which they are found may entitle them to rank with the great mines of the country, such as the Lake copper mines or the Homestake enterprises.

III. Segregated Quartz-Veins.

There are numerous gold-bearing quartz-veins parallel to the bedding, which may be called segregated veins. The gold occurs free in a vitreous quartz, often in quite large pieces, and accompanied with little or no sulphurets. Four men, who have, with a small 5-stamp mill and without capital, operated a vein of this sort during the past year, estimate that they expended altogether 356 days' labor, worth, at $3 per day, $1068. They have had eight clean-ups, which netted them $3383, leaving a profit of $2315. Average samples of their ore assayed at the School of Mines $2.75.

IV. Copper.

Numerous and extensive, but undeveloped, copper-deposits are found in both the eastern and western series of Archæan rocks—the richer apparently being in the latter. They show a copper-stained gossan from 50 to 500 feet wide. This is a surface-accumulation of copper leached from below (?) and deposited near the surface. Where the out-crops or backs of these ore-beds disappear beneath the Potsdam, it, also, is often copper-bearing, clearly deriving this metal as a secondary deposition from the beds below.

At the Blue Lead there is a great quantity of these gossan ores. In sinking through them the miners reached a leached, decomposed portion of the bed, but no one has yet gone far enough to tell what lies below this. Probably after passing the decomposed part, a local enrichment will be found, as in the famous Ducktown deposits, resting upon the unaltered portion below; and beneath this, the ores will consist of pyrite or pyrrhotite, in which there will be mixed a percentage of copper pyrites. The value of these deposits will be directly proportioned to this percentage of chalcopyrite. They will also be found to be distinctly nickeliferous. These points I have determined by tracing the veins down to the creek-beds, where they are less decomposed. They may also be found to carry gold, as the creeks below several of them have yielded gulch-gold.

There is enough of this surface-copper (malachite, red oxide, native copper and copper glance) to justify the erection of a water-jacket smelter at some convenient point. The average tenor of these ores, as assayed at the School of Mines, is about 35 per cent. copper. Such ore would also pay to ship. I know from personal examination that many thousand tons of these surface-ores can be had, and miners now out of employment, by carefully sorting these ores, can make far more than ordinary wages. In their unaltered parts the deposits show all the peculiarities of the Homestake vein, as regards size, carbonaceous matter, structure, cleavage, etc. I regard them as ore-beds of the *Typus Kieslager*, according to the classification of Groddeck.

Ultimately they will be valuable, and will be sought after. At present the surface-ore can be profitably worked. In these gossans the oxide-ores are clearly derived from native copper. This was derived from the sulphide-ores beneath and deposited at the surface as native copper. In breaking a piece of the ore there is often found a nucleus of native copper, surrounded by red oxide, showing

that the latter was derived from the former. When the ores are wholly red oxide, I judge their origin to have been the same.

There is a copper-deposit of a different class upon Box Elder creek. Its value is likewise undetermined. It consists of a belt of lime and magnesian rocks, through which are scattered, in the most irregular manner, galena, chalcopyrite, ilmenite, graphite and pyrite. Chalcopyrite is the most common mineral, and galena probably the second in prominence. The deposit carries gold also in appreciable quantities. The belt of lime and magnesian rocks is said to be 800 to 1000 feet wide. Its economic value needs further study. Its value, if it has any, will probably be found in the gold.

V. Nickel.

This occurs in the beds of pyrrhotite above mentioned. It is always associated with copper. The claims most studied are upon Spring creek. I think I may safely say that there is no body of magnetic pyrites in the Hills which does not carry nickel. The average of our assays is 1½ per cent., though samples carrying 8 per cent. have been found.

VI. Contact-Deposits.

General Description and Extent.—These deposits are found in the Palæozoic rocks in Ruby basin and Bald mountain, at Galena and Carbonate. The former are mainly gold-bearing, while the latter yield silver and lead. They cover a large area—larger than is generally supposed—and must soon become important ore-producers. The ore-bodies are not continuous, but occur in shoots. In the first mentioned districts the shoots have great linear extent. They vary in thickness from a few inches to ten feet or more. Many hundreds of claims showing mineral have been staked and recorded. The value of the ore varies. Some of the ore-bodies will not average over $15, while others exceed $60, per ton. From $20 to $25 may be taken as a fair mean.

These mines are about nine miles from the city of Deadwood, at which place a leaching-plant for the treatment of the ores is now in process of erection.

At Galena the ore-bodies are of equal promise, but of a different character, carrying silver and lead instead of gold. Two smelters have been erected in the camp and run from time to time; but distance from coke and other supplies has heretofore rendered smelting expensive. The near approach of the railroad is believed to have

so reduced the cost that the camp will in future be a steady producer of lead and silver.

These contact-ores are found only in connection with the igneous intrusions. The gold-ores are found in the vicinity of Terry's peak and Bald mountain, where outliers of the Potsdam rocks yet remain. The inference is fair that the rocks once connecting these outliers were also mineral-bearing, and that in the Potsdam, beneath the yet unremoved Carboniferous beds, will be found ores also—at least so far as the igneous intrusions continue. Owing to the durable nature of the upper parts of the Carboniferous limestone, forming, as it does, nearly vertical walls, the underlying Potsdam at its base is usually concealed by a *talus*, making it difficult to prospect; but by many trips down the almost unexplored cañons south of Spearfish, and east and north of Galena, I was enabled to draw a reasonably sure conclusion that the ore-bearing rocks of these sections have a far greater extent than has yet been suspected.

At Galena, the miner sees only the outcrop of these ores. They dip eastward with the enclosing rocks at about 16° below the horizon. After leaving Bear Butte creek, no other streams cut through to the level of the mineralized portions. The questions then occur, How far will these rocks prove ore-bearing? Would a shaft sunk upon the territory lying between the village of Galena and the plains beyond prove it to be ore-bearing?

North and south of Bear Butte creek, there is an area of nearly fifty square miles, where the igneous rocks occur in exactly the same relation to the Potsdam as at Galena. The streams of this area sometimes cut through to the igneous sheets between the Potsdam rocks, enabling one to see this relation, while vertical dikes rise in many places to the surface. I believe that shafts, sunk to the Potsdam anywhere in this section, would be likely to intercept ore. West of Terry's peak and Bald mountain, the same condition occurs throughout a much larger area, extending to the Wyoming side of the Hills, where similar ores are known to exist. In this area a number of streams cut through to the Archæan. Upon a few of these I found ores similar to those of Bald mountain and Ruby basin. Almost any of the small streams, like Cold creek, that flow into Spearfish creek from the east, will show evidence of these deposits, and should be prospected more thoroughly. After a thorough examination, I am convinced that not a tenth of the area in which these ores occur has yet been located.

This section, as well as that east of Bear Butte, is overlain by the

Carboniferous limestone. Where the dikes above referred to cut through this limestone, contact-deposits of a different character occur. They approximate, more or less, a vertical position, and the Iron Hill mine may be taken as a type. The ore-bodies, carrying galena and carbonate of lead clearly derived from galena, are found adjoining the porphyry, where it cuts through the limestone. They differ from the ore-bodies at Galena, in that they are larger, and cut across the bedding, while the former are parallel to it; but they are none the less contact-deposits.

At Iron Hill the Potsdam rocks can be seen, forming the bed of Cold creek; and between them and the Carboniferous limestone, in which the Iron Hill ore-deposit occurs, is an immense horizontal sheet of porphyry, differing in no way in appearance from the vertical dike in the Iron Hill mine. Beneath this sheet, however, the Potsdam rocks are gold-bearing, while the deposits in the immediate overlying limestone are silver- and lead-bearing. It seems as though the same sheet had caused in one series of rocks accumulations of gold-ore, and in the other accumulations of silver-ore. This fact may have a bearing upon the manner in which these ore-deposits are formed. I shall recur to it presently.

Dikes, such as the one connected with the Iron Hill deposits, are common in the area east of Galena, where they likewise cut through the Carboniferous limestone. Adjoining these dikes, is frequently the "liver-colored rock" which formed the gossan or outcrop of the Iron Hill mine. Samples of rock from these gossans, collected by me, assayed from three to ten ounces per ton in silver. I hazard the prediction that ore-bodies similar to the Iron Hill will yet be found there, as well as the continuation of the Galena deposits in the Potsdam beneath, of which I have already spoken.

Before the overlying rocks had been removed, we may imagine that the Galena and Bald mountain sections presented the same appearance as these adjoining areas do to-day; and, had the prospector then been there, he would have seen as little of what lay beneath as he can now see in the area under consideration. Since then, however, the streams have cut through the Archæan rocks at Galena, laying bare, in their course, the outcrop of the lower Potsdam rocks, and enabling us to study the occurrence of ore, from which we should be able to conclude how far the ore-bearing rocks are likely to extend. I venture the opinion that they are co-extensive with the intruded igneous rocks. The question could probably be settled by borings, without the expense of a shaft.

The Ore-Bodies of the Potsdam.—These occur principally along the bedding-planes of the rocks; but deposits are also found occupying the vertical joint-planes, as well as impregnating the quartzite, by replacing the cementing-material of the original sandstone. They are found in such close relation with the igneous rocks as to force the conclusion that they owe their origin to these rocks. At Ruby basin and Bald mountain they usually impregnate distinct zones of the quartzite at different levels in the formation. The most favorable position seems to be upon, or in, the upper parts of the quartzite forming the basal member of the group. Owing to the removal of the overlying rocks, many of the ore-bodies are so thinly covered that the original pyrite has been oxidized. That the iron oxide now forming from 10 to 20 per cent. of the ore-body was derived from pyrite, there seems to be no doubt. In the deeper workings, a peculiar bluish quartzite constitutes the ore. Under the microscope, this is seen to be composed of rounded grains of silica, imbedded in a siliceous paste, which carries very finely divided pyrite, and clearly occupies the spaces between the grains of quartz composing the original sandstone. It was deposited metasomatically, replacing the former calcareous cement; that is, particle by particle, as the original cement was dissolved out, it was replaced by the silica-pyrite cement. The solutions which wrought this change brought also the gold and silver. From the description I have given, it will be seen that these deposits do not, in the ordinary sense, fill pre-existing cavities.

These ores are, in some sections, almost exclusively gold-bearing; in others, they carry partly gold and partly silver, and again in other places the silver predominates. They fill not only the spaces between the grains of quartz, but occur in large bodies, replacing the beds of lime-shales sometimes occurring in the Potsdam. This last is the mode in which the deposits at Galena occur. It seems as though the porphyry at Bald mountain brought mainly gold; at Ruby basin, a few miles distant, gold and silver in nearly equal quantities; while at Galena, twelve miles distant, silver-lead predominates. In the same section the proportion of gold is greater in the lower contact, while in the upper it gives place to silver. That is, broadly speaking, gold predominates in the quartzite, but gives place to silver as we approach the more calcareous portions forming the upper parts of the Potsdam; while in the massive limestone such ore-bodies as are found, like the Iron Hill, carry exclusively lead and silver; yet the porphyry is in all instances the same.

The majority of the conclusions of Mr. Emmons concerning the Leadville deposits will apply to the Black Hills contact-deposits. There is no doubt that they were derived from aqueous solutions, and that they were originally deposited as sulphides. They do not fill pre-existing cavities, and were deposited by a metasomatic interchange. At Galena the ores are mainly argentiferous galena, and the solutions followed by preference certain bedding-planes, which in this case were upon the quartzite and in the lime-shales of the Potsdam. At Ruby basin and Bald mountain the ores were deposited usually in the quartzite itself.

The connection between the overlying porphyries and the ore-deposits is most intimate; where the one is wanting, the other is absent also. As at Leadville, ore is sometimes found without the overlying porphyry sheet, but an examination always reveals the fact that it has been present, but has been eroded.

That the ores were derived from these igneous rocks, is not so plain. At first I accepted the view of Emmons in this particular also, and I am still inclined to hold it so far as the silver-lead deposits are concerned; but I doubt whether all the gold was so derived.

I have described the formation of the Potsdam, how it was built from the ruins of the Archæan, and its contained gold. The intrusion of molten rock, we may believe, was accompanied by or followed by hot waters, and gave rise to thermal springs of long duration. They furnished the hot alkaline solutions which dissolved the gold from the Potsdam and redeposited it in certain favorable localities. This favorable position seems to have been beneath these sheets, and as I have said, in the calcareous portions of the Potsdam, usually at or near the stratification planes of the lime-shales and quartzite. Assays of the most widely separated portions of the Potsdam, and remote from the *locus* of igneous rocks, invariably return appreciable amounts of both gold and silver. Here my results are different from those obtained by Emmons at Leadville. The crystalline schists below have nearly always shown gold. Now the effect of the porphyry sheet may have been only to give rise to the hot solutions necessary to collect and redeposit this gold and silver in an available form. The slates beneath the Potsdam at Galena show, in addition to silver and gold, galena also. This may be seen in Haley's tunnel, and in the Oro Fino mine, which last was clearly formed by solfataric action of the waters coming from below,

and, although not a contact-deposit, will be described here for the bearing it has upon the question now under consideration.

The Oro Fino deposit is found in the Archæan rocks near the overlying contact-deposits of Galena. It consists of a crater or well-like depression, with nearly vertical walls, filled with a breccia which has been already proved to a depth of 235 feet. The cavity has a diameter of 150 feet. The breccia-filling consists of angular and worn fragments of slates, quartzites and porphyries, varying from a few inches to more than a foot in diameter, and firmly cemented together by sulphides, usually pyrite or galena and blende. The pyrite is most abundant, and galena next. A felsite forms one side of the crater. This deposit was formed from solutions from below.

Now the overlying contact-deposits carry the same association of minerals, viz., galena, blende and pyrite. Apparently the solutions which formed the Oro Fino and these deposits are both derived from the same source. As the slates contain many dikes of porphyry similar to the intrusive sheets above (in fact, frequently the same dike cuts indifferently the slates and the Potsdam) the porphyry may have been in both instances the source of the minerals. It is, however, not necessary to assay the porphyry, for both blende and galena can be seen in it in well-defined crystals (never pseudomorphous), while pyrite is seldom, if ever, absent; but whether these minerals were derived from the surrounding rocks and deposited in the porphyry, or *vice versa*, is a question. The evidence points to the view that they were an original constituent of the porphyry and came with it from below, and that all the minerals crystallized from the same magma. Assays of it always show gold. Three distinct "contacts" or levels occur at Galena at which ore is found. They all show practically the same characteristics. There is always a bed of quartzite upon which the ore is found, replacing parts of the thin beds of lime-shales; and over it, but not always in immediate contact, is a sheet of porphyry. The porphyry occurs in thin, irregular sheets, which have been forced between the beds, and which are sometimes seen to be connected with dikes disappearing in the Archæan beneath. No ore-bodies have yet been found to pass from the horizontal deposits into the Archæan, and from their formation I venture the opinion that none will be so found. The Archæan has its own deposits of an entirely different character.

Undoubtedly the ores were originally sulphides. The out-crop of the Galena deposits is almost invariably lead and iron carbonate carry-

ing silver, but the depth at which this changes to sulphide is found to vary with the position of the deposit. In the lower contact, near the creek, the altered portions are soon passed, while in the upper deposits the carbonate is found to continue to very much greater depths, because the upper parts have been much longer exposed to atmospheric influences. In every case where the parts uninfluenced by these agencies have been reached, the iron and lead carbonates give place to pyrite and to galena and blende, thus indicating that these ores originally composed the whole deposit. As at Leadville, the minerals were brought in as sulphides. I believe, also, that they were deposited at great depths; that is, before the Carboniferous, Triassic, Jurassic and Cretaceous rocks which formerly overlay this section were removed. As they were formed subsequent to the intrusion of the igneous rocks, they have been deposited since the close of the Cretaceous, but probably before the beginning of the Miocene, as these rocks were injected and the Hills laid bare to the Archæan during that interval. I do not include the Laramie with the Cretaceous.

VII. Metallurgical Treatment.

The galena ores, of course, offer no difficulty in treatment. At Iron Hill they were smelted without difficulty. In Galena there are two smelters which have run intermittently upon these ores.

The quartzite gold-ores of Ruby basin and Bald mountain seem to have offered real difficulties in treatment. All attempts at working them have heretofore failed. Amalgamation by a modification of the Washoe process was first proposed. After this the use of bromine was tried, with partial success, the loss being, as I am informed, mainly in the silver; but the amount saved was not sufficient to make it as profitable to treat these ores as to ship them to Omaha. The bromine process, as introduced by Mr. Perry Ankeny, seems to have had real merit in it, and should have succeeded.

The subject was then taken up anew by the Dakota School of Mines, where the ordinary Plattner process was tried with fair results, though only in a small way.

VIII. Tin in the Black Hills.

Discovery.—The existence of tin in the Black Hills has been known from the year 1877, when it was recognized by Mr. Richard Pearce of Argo. He determined the black sand in some gulch-gold

sent him from the Black Hills to be cassiterite. The announcement made at the time attracted, however, little or no attention. The practical discovery dates from the year 1883, and the credit undoubtedly belongs to Major A. J. Simmons, of Rapid City, who proved the matter at his own expense. Samples of the ore were sent to San Francisco, and Prof. W. P. Blake was employed to investigate it. A short preliminary notice was made of the discovery and associated minerals in the *American Journal of Science* for September, 1883, by Prof. Blake, who published a more extended notice in the columns of the *Engineering and Mining Journal* during the same month.

General Features.—The known area of tin-bearing rocks has constantly been enlarged, and it now entirely surrounds Harney's peak, and extends into the smaller granite areas lying to the west and south of Custer City, as well as entirely throughout the small Archæan area, west of Deadwood, known as the Nigger Hill district, only a part of which is shown upon the map, for the reason that it extends into Wyoming.

The latter region differs from that of Harney's peak, inasmuch as it contains both the igneous rocks of the Tertiary age and the granitic rocks common to the Harney Peak region. The tin-veins are gold-bearing, the gold probably being carried in the pyrite which sometimes accompanies the cassiterite. This section has been worked continuously for placer gold since its discovery in 1875. The miners have always been troubled with a black sand, which filled the riffles of their sluice-boxes to such an extent as to be a positive nuisance. It was commonly called "iron," and, notwithstanding the note of Mr. Pearce, its true nature seems to have been unsuspected by the miners until after the discovery of tin at the Etta mine. It was then found that the troublesome black sand was tin-ore of good quality. Mr. S. F. Molitor, an assayer of Deadwood, and Mr. Mark Hydliff, of Bear Gulch, both early called attention to this locality. The tin-bearing rocks are partly in Dakota and partly in Wyoming. The whole section is easily reached from Spearfish.

From the manner in which the Potsdam rocks were laid down, there is every reason to suspect, not only that they are, in this area, gold-bearing, but that, since these tin-veins are of Archæan age, the conglomerate forming the base of the Potsdam contains tin also. This did not occur to me at the time of my visit, and I have had no chance to test the question since. As the present streams

have hardly begun to cut into the Archæan rocks of the section, the placer-gold and the large quantities of stream-tin found here can hardly have been freed from the veins by the action of the streams since the Cambrian beds were removed. It is not unlikely that both the gold and the stream-tin, in part at least, were derived from the Potsdam rocks, which of course received both from the great Archæan veins, during the erosion of the slates that took place in Cambrian time.

But little gulch-mining has been done in the Harney Peak region, and, perhaps for this reason, stream-tin has not been found to the same extent as in other sections. Mr. McDermot, the only person engaged in placer-mining in the district, so far as the writer knows, has found in his sluices considerable quantities of stream-tin, a bag of which was taken to the School of Mines, and there run into bars.

The tin-area of the Hills is naturally divided into two localities —the Nigger Hill or Bear Gulch district and the Harney Peak and Custer City districts. The granite rocks of each locality continue to be tin-bearing until they disappear beneath the yet unremoved Palæozoic rocks.

Character of the Veins.—These veins have been classed by Newton and Jenney (who, however, did not observe their stanniferous character), Blake, Vincent and others as igneous or intrusive granites; but they seem to be true veins of the segregated type, parallel to the apparent bedding. Usually they are distinctly lens-shaped, but from a standpoint upon any one lens, a succession of such lenses can be observed upon the line of the strike. The same will doubtless prove true upon the dip. Many of them, however, are tabular in form, and can be traced for thousands of feet. This is true of the First Find, Champion, Tin Reef, Cleveland and many others. The Margaret lode can be traced, with but few interruptions, from Battle creek to Iron creek, a distance of over 6000 feet.

The line of demarcation between the granite vein-matter and the enclosing schists is always sharp and well-defined. There is never any shading or blending of the two. The deposits vary as much in breadth as in length. Some are only a few inches thick, while others exceed 100 feet. The amount of cassiterite also varies. It is never evenly distributed throughout the veins from wall to wall, but lies in zones or sheets. Except their usual parallelism to the bedding, these veins have all the characteristics of true fissures; and two have been observed which cut across the stratification.

The tin-bearing veins differ much among themselves; and even

the same vein often completely changes its character in a short distance. Nearly always, when tin is present, one of the constituents of granite is wanting, and the vein-matter is composed of quartz and mica alone, or of an aggregate of soda-feldspar and mica, which, for want of a better name, has been called *albitic greisen*. At other times the veins consist almost entirely of a massive pinkish feldspar, and again, not infrequently, of quartz alone.

The Etta vein shows some peculiarities.* It is columnar, so to speak, in form. In cross-section it is roughly an oval, having respectively the diameters of 150 and 200 feet. The arrangement of minerals is somewhat concentric. The central portion or core is quartz and feldspar, around which is a zone of albite and mica, carrying tin-stone in considerable quantities. Then comes a zone surrounding this, noted for its large and perfectly formed crystals of spodumene, ten, twenty and even thirty feet in length. The interstices between these crystals are filled with an aggregation of albite also carrying tin-stone, but in a more massive form than the zone just within. Between the spodumene zone and the enclosing mica-schist there is a micaceous aggregate composed of both muscovite and biotite, but barren of cassiterite.

The spodumenes, probably the largest and best-formed crystals of this mineral species ever reported, are not peculiar to the Etta alone, but are found at the Tin Mountain mine, six miles west of Custer City, while smaller crystals are common at many other points in the section. Careful examination has discovered no rule concerning their position with regard to the walls of the lode and to each other. They stand in the midst of the tin-bearing rock, crossing each other in all directions. One was seen over thirty-five feet long, without a break; and, so far as exposed, perfect in form. In other veins, having a more tabular shape, there is usually near one wall a zone of mica, so arranged that the cleavage-planes are either at right-angles or parallel to the walls of the vein. The mica is generally quite fine near the walls, but rapidly becomes coarser until a zone of the largest crystals is passed. The tourmalines were frequently observed to have their vertical axes perpendicular to the wall, but the rule was far from general. Biotite is not common in the tin-veins, and I have never observed it in association with the tin. The prevailing micas are of a light greenish-yellow color, passing into pure white, and are of the common or muscovite species. The feldspar is usually

* First described by Prof. W. P. Blake.

albite and the variety known as cleavelandite is common. Lithia-mica has not been observed, notwithstanding lithia is found in such quantity as is evidenced by the spodumenes.

It is, perhaps, worthy of remark that when the three elements of granite are together in anything like ordinary proportion, tin-stone is very likely to be absent. When the aggregation consists mainly of mica and quartz or of mica and albite, tin is usually present also. If the vein consists of feldspar alone, the tin is wanting; but when it is composed of quartz alone, as is frequently the case, the quartz is always banded as in a fissure-vein, and is usually tin-bearing; but the tin-stone is quite different in appearance from that found in the greisens. It is light reddish-brown, and often occurs at the joints or planes indicating the banded structure. The crystals of cassiterite usually partake of the nature of the individual crystals composing the rock. If they are large, the tin crystals are large also, and *vice versa*. If the rock is quartz and mica, neither the crystals of quartz nor of mica enclose the crystals of cassiterite. They occupy interstices, so to speak, between these crystals; but in the albitic greisen they are frequently enclosed in feldspar. In the quartz-veins, the quartz frequently encloses the tin-stone.

The abundance of phosphatic minerals present is an extraordinary feature of these veins. Apatite, triphylite and heterosite are found in large quantities; autunite occurs sparingly; and other phosphatic minerals, perhaps new, but needing further examination, are also found.

Of other associated minerals, columbite occurs in large quantities. Masses many pounds in weight are common. The Bob Ingersoll claim is reported to have yielded a mass* weighing a ton. This mineral is here frequently called tantalite; but all analyses so far made for me at the Dakota School of Mines, as well as a great number of specific-gravity tests, have invariably shown that it is columbite. Tantalic acid is always present in varying quantities, but never to such a degree as to be the predominating acid. The crystal form is always tabular, while the *habitus* of tantalite is prismatic.† A sample, however, from the Etta mine, analyzed by Professor Schaeffer, was pronounced by him to be tantalite.‡ It gave the following result:

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* W. P. Blake.

† Naumann's *Mineralogie*, 1874, pages 523 and 525.

‡ *Trans.*, xiii., 231.

	Per cent.
Tantalic oxide,	79.01
Stannic oxide,	0.39
Ferrous oxide,	8.33
Manganous oxide,	12.13
Specific gravity, 7.72.	99.86

Professor Schaeffer further declares that he was unable to find the least trace of columbic acid.

The following analysis of a sample taken from the Etta mine was made at the Dakota School of Mines, by Dr. W. P. Headden, and gave:

	Per cent.
Tantalic acid,	18.20
Columbic acid,	64.09
Ferrous oxide,	11.21
Manganous oxide,	7.07
Stannic oxide,	0.10
Calcic oxide,	0.21
Specific gravity, 5.89.	100.88

Five other samples from different sources—some from stream-tin, others from lodes—were examined, and all gave very decided tests for columbic acid, and ranged in specific gravity from 5.89 to 6.12. By these results we have been led to the conclusion that if tantalite —that is, a mineral composed mainly of tantalic acid—exists in the Hills, it is comparatively rare.

Graphite, both amorphous and crystalline, is common in these veins. It is sometimes very pure. Galena, carrying silver, is occasionally met with, while arsenopyrite, carrying gold, is not uncommon. Beryls are found in all the veins, and often of large size. Garnets exist in all mines so far worked. Two varieties were observed, the ordinary iron garnet and a honey-yellow garnet, quite common, but very small, supposed to be a lime-garnet. Barytes is quite common. Ilmenite I have observed. Zircons and corundum have been reported by others. It is clear that the veins will form an interesting study for the mineralogist, as we have every reason to believe that new mineral species will be discovered here. The minerals so far observed by me in these veins contain the rare elements — boron, phosphorus, fluorine, tin, lithium, glucinum, uranium, tantalum and columbium, as well as the more common ones—gold, silver, lead, arsenic, iron, sulphur, etc.

I have been thus explicit in describing these veins in order to

show how perfectly they fall into the class of certain granitic veins of the Laurentian, described by Dr. Hunt. These are the endogenous veins which, he says,* "have been deposited from solutions in fissures of the strata, precisely like metalliferous lodes." This remark applies especially to those granitic veins, which include minerals containing the rarer elements. Among these are boron, calcium, rubidium, glucinum, zirconium, tin and columbium. Much of the remaining evidence which Dr. Hunt cites in support of his theory of the aqueous origin of certain granites in the Laurentian, and of the distinction he draws between granitic dikes and granitic vein-stones, is furnished here also: for example, the banded arrangement of minerals, the peculiar manner in which certain minerals incrust one another, and the rounded form of certain crystals.

The vein-stones of these lodes vary but little in ultimate constituents from the surrounding schists, and were probably derived immediately from them by circulating waters. The proportion of alkalies in the vein-stones is 15 per cent., while that in the schists is 8 per cent.† Many of the elements found in the veins are known to exist in the schists, viz., gold, silver, sulphur, iron and carbon (graphite). The rarer elements, such as lithium, columbium and tin, which probably exist in the great mass of sediments in the most minute quantities only,‡ are found in these veins in surprisingly large amounts. This accumulation is best accounted for upon the supposition that they have been collected and deposited by percolating waters.

As neither the veins nor the large bodies of granite seem to be either eruptive or metamorphic in origin, I conclude that such granites are not found in the limits of the Black Hills.

In the Cleveland vein, in Bear Gulch district, parts of the granite vein have been altered to a fine-grained, dark green, almost black, rock. Its boundaries in the vein are as sharply defined from the white-colored granite which they traverse as the granite is from the enclosing mica-schists. It carries a little fine-grained cassiterite and is believed to resemble closely the *Zwitter* of Altenberg. A further examination of it is now in progress.

Daubrée has stated that the presence of tin is intimately connected with quartz, and that, after quartz, certain minerals not com-

* *Chemical and Geological Essays*, page 192.

† Newton; but this can hardly be correct; for solid feldspar would contain but 15 per cent.

‡ Dr. Hunt.

mon to ordinary lodes are habitually present. These are silicates into which fluorine and boron enter, among which are tourmaline, certain micas, lepidolite, topaz, with apatite and other phosphates. He supposes that the tin was introduced into the lodes originally as a fluoride or boride, or possibly in combination with chlorine or phosphorus.* In the Black Hills veins, lepidolite and topaz do not occur. Tin is often found remote from quartz. When tourmaline greatly predominates, the tin is absent.† The greatest quantities of cassiterite occur in veins consisting of mica and quartz alone. The micas seem always to be ordinary muscovite. They have not been tested for fluorine. Fluorspar has not been found. No pseudomorphs of cassiterite, as in Cornish veins, have been observed. Orthoclase occurs very sparingly; the variety, microcline, much more frequently. Albite is everywhere the predominating feldspar.

Analyses of Cassiterite from the Veins.—The following analyses of Black Hills cassiterite were made for me by Dr. W. P. Headden:

No. 1. Occidental mine:

Stannic oxide, ...	96.42 = 75.86 tin.
Insoluble, ...	3.58
Sp. gr., 6.923.	100.00

No. 2. Tin Mountain mine:

Stannic oxide, ...	97.5 = 76.7 tin.
Sp. gr., 6.923.	

No. 3. First Find mine:

Stannic oxide, ...	94.70 = 74 5 tin.
Sp. gr., 6.728.	

For the purpose of comparison I append the following analyses of tin-ore from other sections:

Locality.	Stannic oxide. Per cent.	Sp. gr.	Authority.
Finbo, Sweden, . . .	93.6	——	Berzelius.
Wicklow, Ireland, . .	95.26	6.75	Mallet.
Xeres, Mexico, . . .	89.43	6.862	Bergemann.
Tipuani, Bolivia, . .	91.80	7.021	Forbes.
Zinnwald, ...	88.04	——	
California, ...	76.15	——	Genth.
Cornwall, ...	98.93	——	Klaproth.
Schlackenwald, . . .	99.28	——	Klaproth.

* Prestwich, vol. i., page 344.

† That is, in the veins; but since the above was written we have found cassiterite in the schorlaceous schists themselves, which sometimes form the enclosing rocks of the veins. It was found, however, quite close to the veins.

I give also an analysis of Cornish black tin ready for the market:*

	Per cent.
Stannic oxide,	92.00 = 72.37 tin.
Ferric oxide,	4.32
Insoluble,	1.66
Water,	2.00
	99.98

Analyses of Stream-tin.—A peculiar fact has been observed in regard to the stream-tin. It is clearly derived from the disintegration of the tin-veins, but it is not so pure as that which remains in the veins, as the following analyses by Dr Headden will show:

No. 1. Nigger Hill stream-tin:

	Per cent.
Stannic oxide,	92.60 = 72.84 tin.

No. 2. Southern Hills stream-tin:

Stannic oxide,	92.80 = 73.0 tin.

No. 3. Nigger Hill stream-tin:

	Per cent.
Stannic oxide,	93.00 = 73.21 tin.
Iron,	2.16
Insoluble,	3.96
	99.12
Sp. gr. (average of several samples),	7.

Is it possible that a part of the almost insoluble cassiterite has been dissolved out and replaced by some other mineral?

Percentage of Black-Tin in the Rock.—This question has been much discussed. If all the stuff broken at the mine were sent to the mill, the percentage of tin would be low. By close sorting it could be made very high, depending entirely upon the extent to which the lower grades of ore were rejected. Assays signify nothing. Hand-samples containing from 25 to 50 per cent of black-tin can be readily selected at any mine. Professor W. P. Blake suggests that it will probably be found advisable to send to the mill for treatment all rock carrying per ton 10 pounds of cassiterite and upwards. If this be done, I estimate 2 per cent. cassiterite as the probable average of the tin-ores of the Hills thus treated.

* Taken from Moissenet's *Préparation Mécanique du Minerai d'étain dans le Cornwall*, Paris, 1858, page 204. (Reprint from *Annales des Mines*, 5e ser., tome xiv. p. 77.)

Treatment of the Ore.—In the treatment of the tin-ore, difficulties were anticipated which were not subsequently encountered. The first experimenters devised ingenious methods for ridding the ore of the large mica crystals before attempting to concentrate it. They also used either the Frue vanner or a shaking-table of some sort for dressing the ore. It seemed to be taken for granted that ordinary jigs and buddles would not do the work. Investigation, however, showed that owing to the crystalline nature of the rock, and the coarseness of the crystals of cassiterite, a good separation could be had with a coarse crushing, and that it made an ideal jig-product. It was determined, therefore, to erect jigs for its treatment in the concentration-plant at the School of Mines laboratory. The following is from the official report upon the first test made there, which was conducted by Professor H. O. Hofman and myself:

The crushing and concentration plant at the School of Mines, upon which the test was made, consists of a small jaw-crusher, one pair of 14-inch Cornish rolls, four trommels or revolving screens and three Harz jigs, with eccentric motion. It will be observed by engineers that there is no provision for either fine crushing or fine separation, and that the plant is imperfect to this extent. It was the intention of the management to have added an Evans buddle, but the funds were exhausted before this was reached. It is true the mill contains stamps and a Frue vanner, but they were not used in this test.

The ore was of greisen, but, unlike the typical greisen of Europe, it is extremely coarse grained, and could not, from the large crystals of mica, be stamped. It was crushed and carefully assayed, and gave a return of 2.7 per cent. black tin. It is believed that it represented just such material as a mill, upon a commercial basis, would have to treat in the tin section. It was first put through the crusher, and then through the rolls above mentioned. No attempt whatever was made to treat the jig tailings, which were quite coarse, by recrushing and further concentration, in order to save such fine tin as they might yet carry. The screens upon the trommels were, respectively, fourteen, fifteen, and thirty mesh. That which did not pass through the first, or coarse screen, was returned to the rolls. Forty pounds of coarse stream-tin was used to bed the first compartment of the first jig, and one jig only was used, as there were but 1800 pounds of the material to be tested. The products from the screens were treated in the order of their coarseness, beginning with the coarsest, and taking one grade after the other, but using always the same jig which made 120 lifts of 1-inch per minute. The number of revolutions per minute could not be changed, but the lift was lessened to about one-half inch for the last product. The tailings were constantly sampled as they left the jig by many competent persons, and only occasionally was a trace of tin observed. This shows, that so far as the tin had been separated from the gangue by crushing—just so far the separation was perfect.

In a mill running with a commercial purpose, these tailings would probably be crushed still finer, that a more perfect separation of fine tin might be had. We supposed it would be necessary in this case, but a careful assay of the tailings disclosed the fact that 95.15 per cent. had been saved—a result as extraordinary as it was unexpected. It will be observed that there yet remain in the tailings 2.08

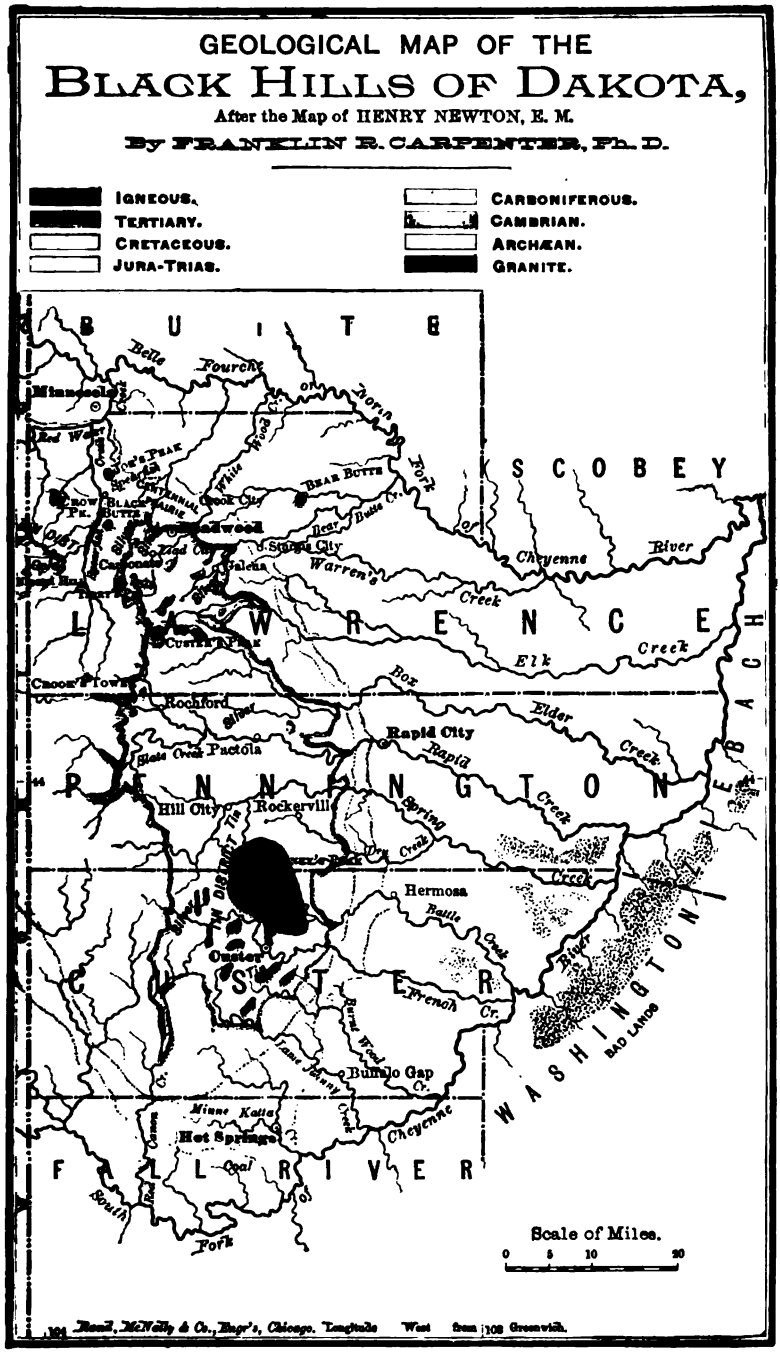
pounds per ton of black tin; but in Cornwall, for every ton of tin-stuff treated on the dressing-floors, discharging their waste into Red River, there are returned by the stream-works upon the river never less than 3 pounds of tin. It has, in some years, reached 6 pounds for each ton treated. As heretofore stated, the cost in Cornwall, notwithstanding their cheap and skilled labor, is $1.25 per ton of tin-stuff dressed or concentrated. I feel sure that ours can be dressed at a cost not exceeding 50 cents per ton. This difference in cost is owing wholly to the difference in the character of the ores, ours being suitable for coarse concentration, while theirs must be stamped very fine, and hence cannot be jigged to any advantage, so that little or no concentrating machinery is used, nearly the entire work being done by hand. This necessitates the employment of nearly as many operatives in the dressing-works as are employed in the mines. It is true, that many of them are boys and girls, but their number is sufficient to raise the cost of dressing the tin-rock to $1.25 per ton. (For detailed information see the paper of Mr. R. J. Frechville, extracts from which appeared in Mineral Resources, United States Geological Survey for 1885.)

The concentrates from this test returned 68 per cent. tin. A sample, however, made entirely free from gangue, showed by analysis 74.50 per cent. tin. Other tests have been made since, upon large lots in which all the jigs, etc., were employed as in a working mill. One lot of several tons, containing less than one-half per cent. black tin, was so treated that the loss was even less than in the test given in detail above. We conclude that the ore can be crushed without difficulty. Less loss and a better separation would probably be had by doing the entire crushing by jaw-crushers, multiple or otherwise, than by rolls. We found that the work might be expedited by screening out the coarsest of the mica, and used for this purpose a screen of two meshes to the linear inch. This, however, is not absolutely necessary; but as this product is found to be practically free from tin, often one-third of the waste may be thus removed without further trouble. In the above test we afterward jigged the screened mica, more to see its behavior in a jig than for what tin it might contain, and found no difficulty whatever in its management.

We unhesitatingly recommend jigs for the treatment of the Black Hills tin-ores.

Yield of Metallic Tin.—It may be set down as a fact, that the clean cassiterite of the Black Hills tin-veins will not, by analysis, fall below 74 per cent. tin, and when perfectly clean it will yield a white tin of the grade known to the English tin trade as "*superior refined*" quality. This it does without the previous roasting that is required for most of the Cornish ore. It is of the utmost necessity, however, that the black-tin be clean. There is a heavy brown garnet occurring with the tin, which can be separated by careful washing, and which must be removed in order to obtain good results. Strangers testing these ores for the first time are apt to leave it with the cassiterite, and hence bad results have sometimes been reported when good results would have been obtained by proper manipulation.

Recent tests of satisfactory character, made by English and other experts, for the Harney Peak Company, are omitted here, in order that this paper may not be still further extended.


GEOLOGICAL MAP OF THE
BLACK HILLS OF DAKOTA,
After the Map of HENRY NEWTON, E. M.
By FRANKLIN R. CARPENTER, Ph. D.
IGNEOUS.
TERTIARY.
CRETACEOUS.
JURA-TRIAS.
CARBONIFEROUS.
CAMBRIAN.
ARCHÆAN.
GRANITE.
BUTTE
SCOBEY
LAWRENCE
PENNINGTON
CUSTER
FALL RIVER
WASHINGTON
BEADLE
Bella Fourche
Belle Fourche
Minnesela
Bear Butte
Cheyenne River
Warren's
Creek
Elk Creek
Box Elder Creek
Rapid City
Rapid Creek
Spring Creek
Rockford
Pactola
Hill City
Rockerville
Hermosa
Battle Creek
Custer
Buffalo Gap
Hot Springs
Cheyenne
South Fork
Scale of Miles.
0 5 10 20

THE WENSTRÖM MAGNETIC SEPARATOR.

BY ROBERT ANDERSON COOK, A.M., NEW BRUNSWICK, N. J.

(New York Meeting, February, 1889.)

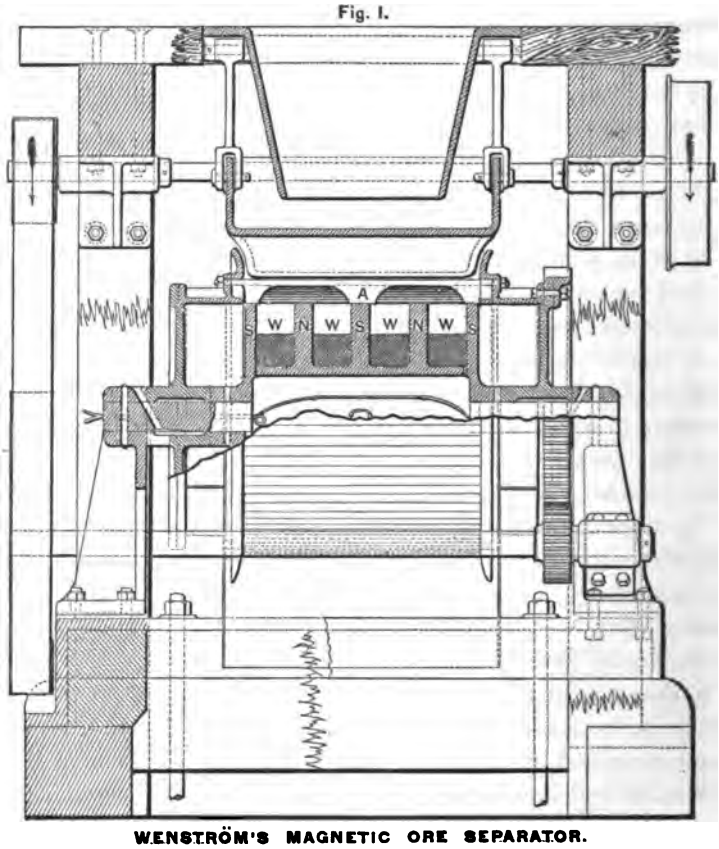
As the margin of profit in the manufacture of iron continues to decrease, attention is called more and more to economies in every department. Beginning at the bottom, in the preparation of the ore at the mine, we notice a general tendency is towards the shipment of richer ores, securing to the mine-owner a better price per ton, which the purchaser is glad to pay by reason of the saving in freight of iron, and in the cost of reduction, which depends so largely upon the amount of fuel and flux, and the productive capacity of the plant. In other words, the difference in value between rich and poor ores is much greater than their relative percentages of iron indicates; because the barren material in a lean ore is not merely worthless, it is a positive source of expense, requiring transportation, handling, fluxing, smelting, and its due share of general expenses; in return for which it yields nothing and decreases the available capacity of a given plant and capital for profitable work.

The enriching of the ore is done in most places by hand-picking, and in a few by wet concentration; but the managers of several magnetite mines in Sweden have been practically successful in doing it by machinery. The only magnetic separating machine which has actually taken the place of hand-picking was designed and patented by Mr. Jonas Wenström of Örebrö, Sweden.

In this machine, which is illustrated in Figs. 1 and 2, Mr. Wenström has taken advantage of the property of soft iron to become magnetized by induction. The machine consists of a barrel made up of soft iron bars and some non-conducting material (usually wood), which is rotated around a stationary shaft by means of a cog-wheel. Inside of this barrel, on the shaft, is placed eccentrically a cylindrical electro-magnet, provided with a number of flanges (N, S, Fig. 1), between which wire is wound in such a manner that the flanges are of negative and positive polarity, being energized by a current passing through the wire coils from a small dynamo. As will be seen in Fig. 1, the flanges, N, S, are circular, and follow the internal contour of the barrel on the front side, while on the opposite side they are cut down to the line B, B′ in Fig. 2. Hence the bars of the revolving barrel on the front side of the machine are mag-

netic only while passing from B to B′; and after they pass the point B′ in the lower part of the machine, they cease to be so, and remain entirely without magnetism in passing through the space from B′ to B, at the back of the machine. The bars have alternately two and three projections coming close to the polar flanges, so that each bar becomes virtually a prolongation of the magnetic poles.

Fig. I.



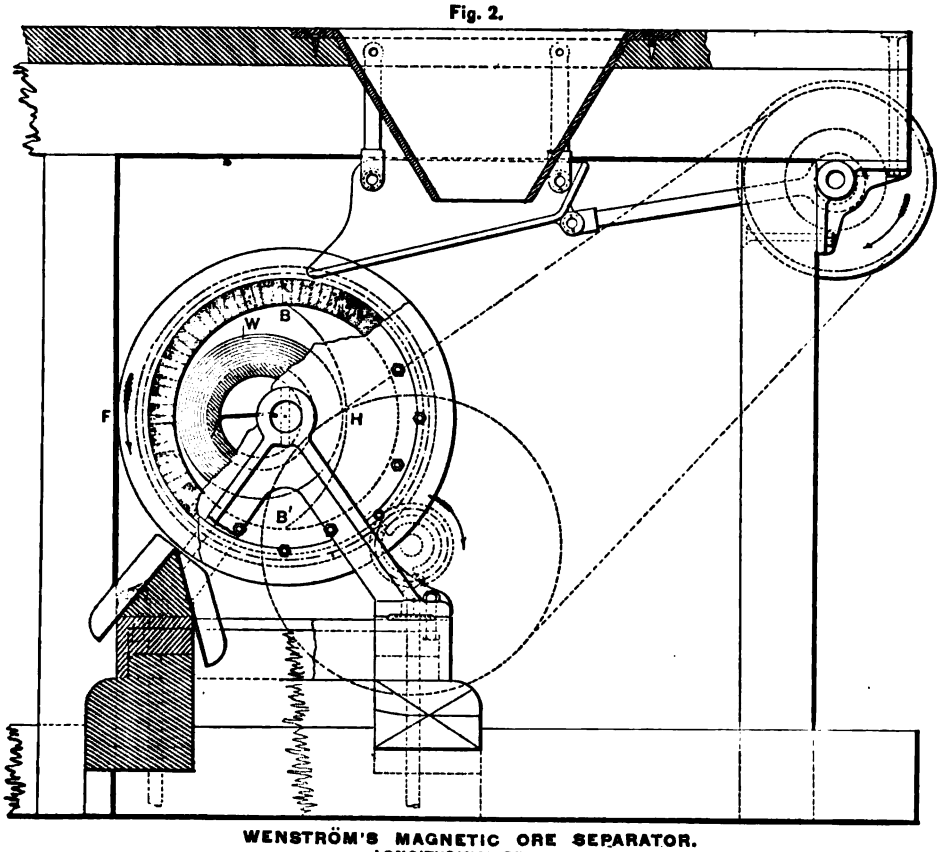
WENSTRÖM'S MAGNETIC ORE SEPARATOR.
CROSS SECTION
Scale: 1 in.=1 foot.

The ore and rock are fed on top of the revolving barrel, coarse and fine together, and the ore adheres to the barrel during half a revolution, being only released after it has passed out of the magnetic field at B′; the non-magnetic material rolls off the barrel and falls in front of the dividing-board, while the magnetic material drops off behind it.

Two sizes of this separator have been introduced. The largest size

treats from six to seven tons of material per hour, and the magnetism is strong enough to support pieces of ore up to seven pounds in weight, and separate them from the rock. The smaller size treats from two to three tons per hour of finer material, below three-quarters of a pound in weight of single pieces.

Fig. 2.



WENSTRÖM'S MAGNETIC ORE SEPARATOR.
LONGITUDINAL SECTION
Scale; 1 in.=1 foot.

In Sweden these separators are used at the iron-mines for extracting from old and new dumps of waste material the good ore which has been missed in the hand-picking, or was too fine to be picked out in that way. They are also applied to the ore now being mined, which in former times was selected by hand. Three of the Swedish mines use the larger size of the machine, and four the smaller.

During a recent journey in Sweden (unfortunately in the winter, when the dumps were frozen and the machines idle), the writer

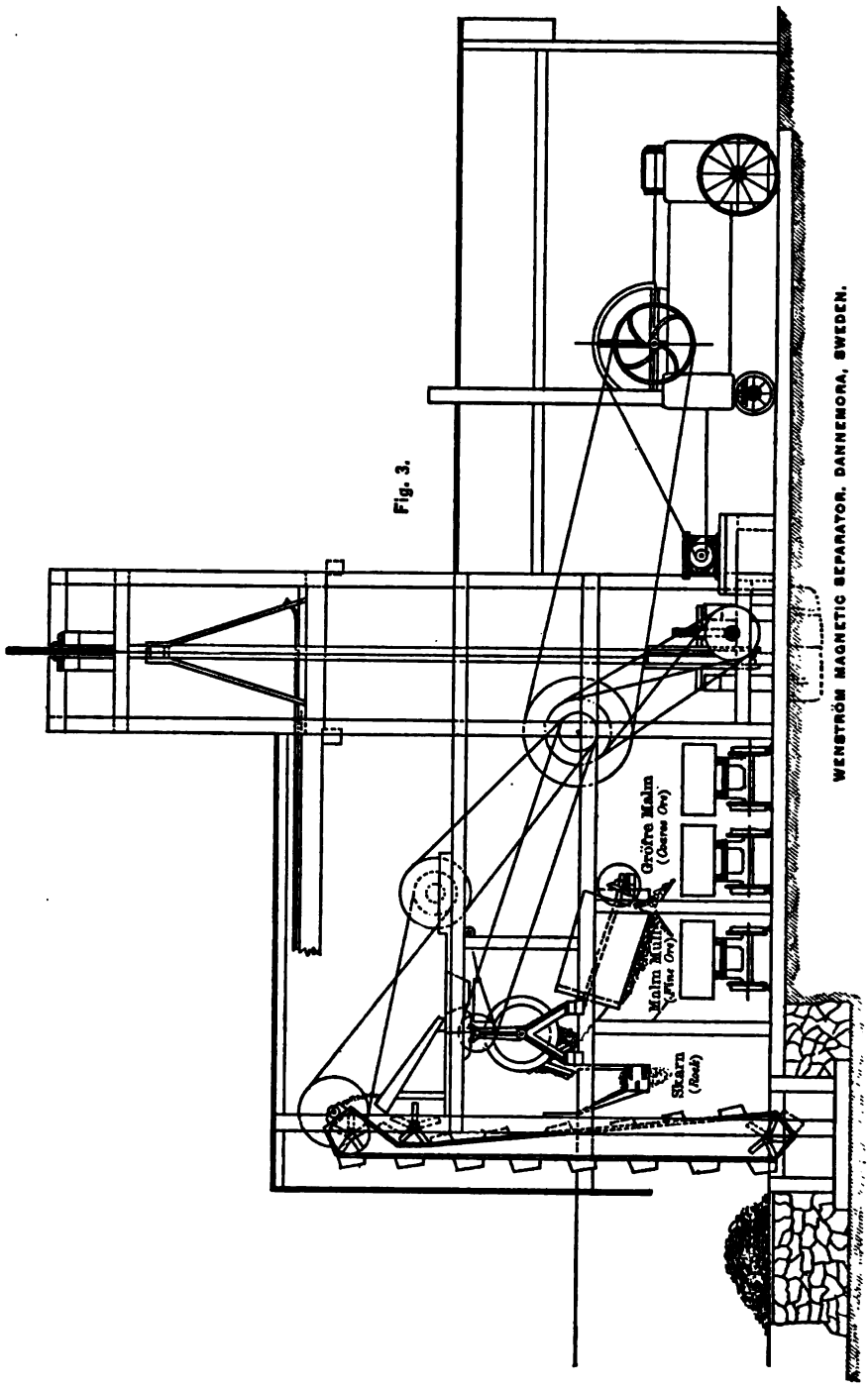


Fig. 3.

WENSTRÖM MAGNETIC SEPARATOR. DANNEMORA, SWEDEN.

visited five of the mines where these machines are used, and obtained the records of their working-results. It is deemed sufficient to give in this paper, as a typical example, the data thus collected at the Dannemora mine. Practice elsewhere is essentially similar, the cost varying at each mine in proportion to the handling necessarily given the ore before it comes to the separator.

The Dannemora plant (Fig. 3) consists of a separator, 24 inches on the face and 27 inches in diameter, a small dynamo, and a portable engine, which runs not only the separator but also a bucket-elevator and a hoist capable of raising a car loaded with half a ton of ore. The material, which is anything below a four-inch mesh, is dumped from a trestle at the elevator, which conveys the material to the feed of the separator, and the separated rock falls into a chute, and is thence delivered into a car, and run off to the waste dump. The ore runs down into a barrel-screen of perforated steel, the holes being one inch in diameter. The pieces which pass through fall into one car, while the coarser portion fall into another. These cars are then hoisted on the elevator and run off to the point of shipment. The fine stuff, below one-inch mesh, is kept by itself for reasons connected with the subsequent roasting. All the ore is roasted, and there is some difficulty in roasting the fine ore. Consequently not more than 5 or 10 per cent. of the charge in the roasting-kilns is permitted to be fine. The fuel used in the kiln is the tunnel-head gas from the blast-furnace.

Four laborers and an engineer are required to run the machine to its full capacity, with a boy to pick out from the rock any pieces of ore which may have been mechanically knocked off by larger pieces of rock. The men's work consists of shoveling the ore into the elevator, and taking the rock and ore in the cars to the dumps. The amount of material handled averages six and a half tons per hour.

The separator was introduced in 1886, during which year no permanent plant was erected, but the machine was moved around from one small waste-heap to another. The present plant (Fig. 3) was erected in 1887, and during that year, as also in 1888, the separator was run intermittently through a season of five months. The work of the three years was as follows:

Year.	Total material treated. Tons.	Ore obtained. Tons.	Percentage of total material obtained as ore.	Rock Tons.
1886	2000	No data	—	——
1887	5720	3609	63	2111
1888	5169	3925	76	1244

The cost of hand-picking at Dannemora, previous to the introduction of this machine, was 32.6 cents per ton. It now costs 9.8 cents per ton,* and 30 per cent. more ore is obtained from the same material. Analyses of these ores are not frequently made, but it is known that the separated ore averages 59 per cent. iron for the coarse, and 45 per cent. for the fine. If the fine ore can be fed by itself in a somewhat different way, the results, as to that part, would be much better.

Any invention adopted from a foreign country, where the plans of working, as well as the ore treated, are different from our own, must necessarily be changed to some extent to do the work required in America. Apart from this consideration, each ore must be treated in a different way. But that this invention is of practical value to us in America is demonstrated by the fact that it is used profitably in Sweden, where labor is not worth more than fifty cents per day.

There are three different cases to which this machine can be successfully applied, with only one change in the feed (namely, when the ore is below a half-inch mesh). First, where there are waste-dumps, and the run of mine has to be hand-picked. Second, where only what might be called preliminary crushing is necessary to break the greater part of the ore free from the rock, that is, by passing it through an ordinary crusher to a half-inch mesh—which ought not to cost more than fifteen cents per ton. Third, where the ore must be pulverized by further machinery to a one-quarter-inch mesh or some smaller size.

The first two of these cases are very simple, but the third is a problem by itself, in which the cost of mining, the number of tons which must be crushed to obtain one ton of concentrate, and the cost of crushing, are items of far more importance financially than the mere separation of the ore from the rock. Each mine-owner must decide for himself whether it will pay. The cost of separating ore per ton of finished product by any machine, it is impossible to give, since it varies with the richness of the ore. A Wenström machine requiring one and a half horse-power to run the dynamo, and half a horse-power, or less, to run the separator, has a capacity of fifty tons of fine (or a larger amount of coarse) material in ten hours. The feeding should be automatic, and the ore and rock should fall by their own weight into cars, so that there need be no hand-labor,

* This includes interest on plant and depreciation. The cost for repairs has been nothing, and the machine shows no wear.

except that of feeding the crusher or screen, if coarse material is to be used.

Some interesting questions arise in connection with the concentrated fine ore.

One, which the Swedish government settled to the satisfaction of its furnace-manager, is that of the use of fine ore in the furnace. It had always been supposed to be a detriment to the furnace to charge a large proportion of fine ore. Experiments were made at the expense of the government, under the charge of Mr. Granstrom, of Norberg, an eminent Swedish engineer, in the preparation of briquettes of a mixture of fine hematites crushed and concentrated to 65 per cent. of iron, with slacked lime, in a powerful brick-machine. These briquettes were very nice-looking; but, when put in the furnace, they all went to pieces. Mr. Granstrom then resolved to see exactly how much fine ore a furnace would stand. Taking one of the charcoal furnaces in his charge, he began by using a mixture of 10 per cent. fine ore and gradually worked it up to a maximum of 75 per cent. The only change he found with the latter mixture was that the gases were a little hotter, and he was able to carry a little heavier burden on the furnace. The pressure was the same; and the conclusion reached was that the manufacture of briquettes was entirely unnecessary.

Another question concerns the effect of fine concentration in reducing such impurities of the ore as enter the pig-iron made from it. This has never yet been carried so far in practice as to convert a non-Bessemer ore into a Bessemer ore, by eliminating, through the process of concentration, the phosphorus contained in the gangue; but in all the concentrations of ores, wet or dry, the impurities have been materially reduced; and it seems not unreasonable to expect that many ores, which are but little outside the "Bessemer limit," will be brought within that limit, and thus made more valuable.

There are other uses to which this magnetic separator has been profitably applied in Sweden. One is that of separating the pieces of iron from the dirt, in yards where pig-iron has been stored, or from any old heaps of waste material—for instance, where the dumpings from cupolas have been piled.

One of the small Wenström machines was brought over from Sweden to this country last year, and tested with very satisfactory results on limited quantities of different ores and waste material from around steel-works. One instance in particular may be mentioned, namely, the treatment of cinder from the Bessemer con-

verter, a material which is almost wholly thrown away at American steel-works, a small portion only being used in the blast-furnaces. A hundred pounds of this material, crushed in an ordinary Blake crusher, was run over the separator, the result being twenty-five pounds of magnetic material containing 70 per cent. of metallic iron. The same quantity of this waste product from another steel-works gave thirty-eight pounds of magnetic material.

A machine was sent, a short time ago, to the Lackawanna Iron and Coal Company, at Scranton, Pa. At these works the waste from the cupolas, and from around the runners at the converter, has been for a long time treated by putting it in a "rumbler," such as is used for cleaning castings at foundries, only of a larger size. The fine pieces of slag and iron are thus broken up and fall through the rumbler; and these have always been sent to the waste dump, while the coarser pieces are taken from the rumbler and sent to the cupola to be melted over again. At present, the fine waste, before it goes to the dump, is put over the Wenström separator.

The following is the result of eleven days' running (from March 3 to 13, 1889), four and a half hours per day, 198½ tons of waste being put over the machine. From this quantity there was obtained 35 tons, 3 cwt. of iron. Two men were added to the force at the rumbler to handle the material to and from the separator. The engineer was the same who ran the rumbler, and the steam for running the separator was obtained from the same boiler; so that the total expense was only two dollars and a half per day for coal, labor and oil.

The handling of converter-cinder on a large scale has not been undertaken as yet by the Lackawanna Company, the crusher required not being ready.

NOTES ON THE BERNICE ANTHRACITE COAL-BASIN, SULLIVAN COUNTY, PA.

BY CLARENCE R. CLAGHORN, BIRMINGHAM, ALA.

(New York Meeting, February, 1889.)

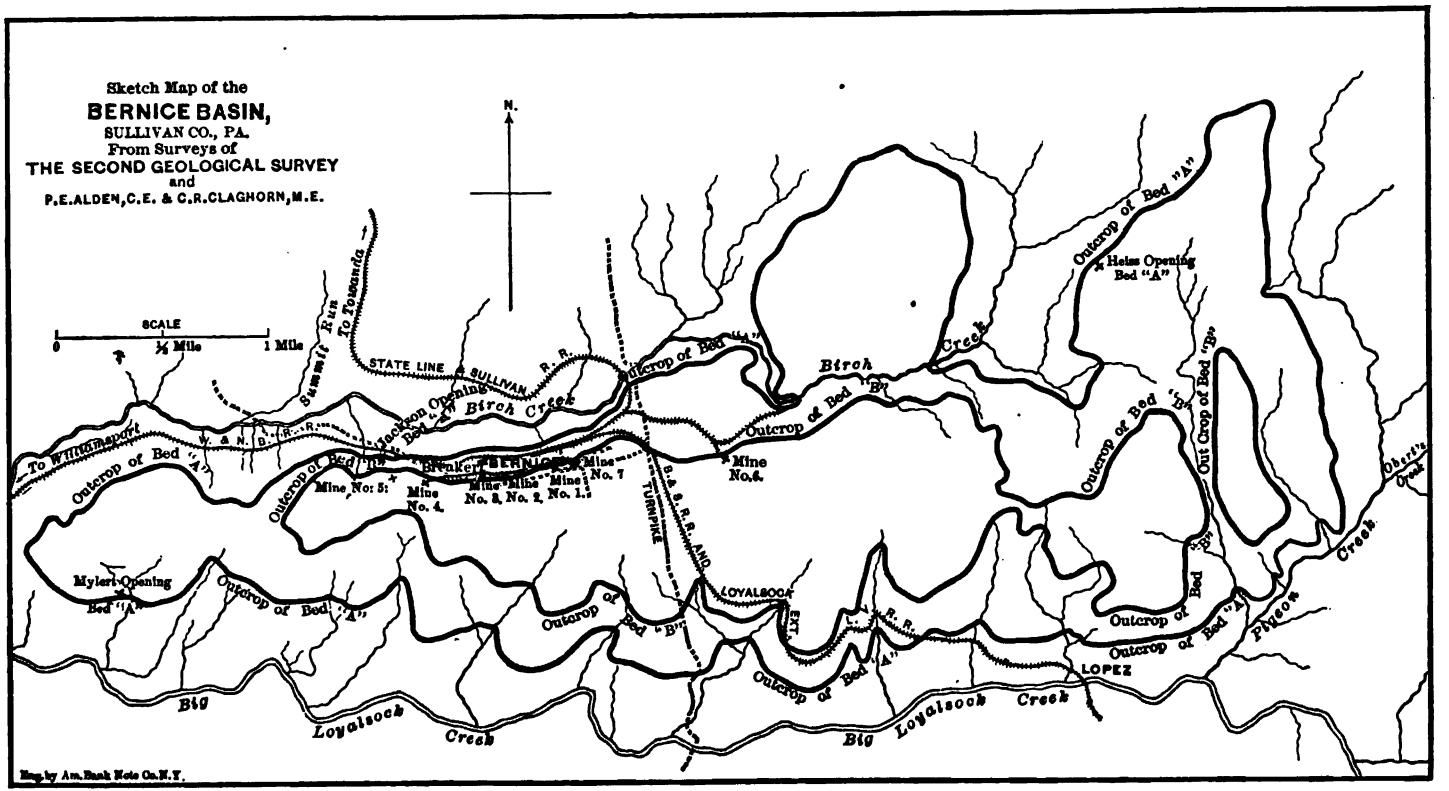
FEW facts relating to the Bernice anthracite coal-basin have been published in our *Transactions*, and little is known among engineers in general of the character and composition of the coal found in that basin.

The Bernice basin is the westernmost and largest of a number of comparatively small coal-basins in Sullivan and Wyoming counties, Pennsylvania, formerly known as the Loyalsock Mahoopany coal-fields, but now collectively designated in the reports of the Second Geological Survey of Pennsylvania as the Western-Northern anthracite-field, on account of its geographical position with respect to the other fields of the anthracite-region. The Bernice basin is situated in the eastern part of Sullivan county, on the North or Allegheny Mountain plateau (2000 feet above tide), in the dividing ridge between Birch creek on the north and the Big Loyalsock creek (after which the basin and the coal were named) on the south, the drainage of the basin being divided between these two creeks. The topography of the region underlain by coal is irregular, but comparatively flat; and the underlying rocks conform in the main to the configuration of the overlying ground. This, coupled with the fact that the whole region is exceedingly wild and totally undeveloped, being still covered with virgin forests of hemlock, beech, birch and maple, renders it impossible to obtain an accurate geological section of the measures, or to do more than approximately locate the outcrops of the included coal-beds. In 1883 the Second Geological Survey of Pennsylvania, in conjunction with the State Line and Sullivan Railroad Company, made a topographical map, in ten-foot contour-lines, of the area included within the basin; and in 1884, Mr. Chas. A. Ashburner, geologist in charge of the anthracite-district of the Pennsylvania Survey, made a geological exploration of the field and located, as nearly as was possible, considering the character of the country, the limits of the coal-beds.*

While resident-engineer at the mines at Bernice, I made a number of geological sections of the measures of the basin; but they were very conflicting and unsatisfactory, because few rock-exposures were to be had. However, the sections made by Mr. Platt in 1879,† and those of Mr. Ashburner made in 1884, agree closely with mine, and indicate that the red rocks of the Mauch Chunk period (No. XI. of the Pennsylvania series) come in not more than 180 feet below the coal. In the Northern anthracite-field, only forty miles away, the Pottsville conglomerate, No. XII., has been proved conclusively to be very much thicker than this; and this fact, together with others observed

* The results of Mr. Ashburner's careful examination of the field, together with the topographical and geological map already referred to, are contained in the *Annual Report*, for 1885, of the Second Geological Survey of Pennsylvania.

† Report GG., *Lycoming and Sullivan Cos.*, Penna. Geol. Survey.


Sketch Map of the
BERNICE BASIN,
SULLIVAN CO., PA.
From Surveys of
THE SECOND GEOLOGICAL SURVEY
and
P.E.ALDEN, C.E. & C.R.CLAGHORN, M.E.
N.
SCALE
0
½ Mile
1 Mile
Summit Run To Towanda
STATE LINE
SULLIVAN
R. R.
Outcrop of Bed "A"
Birch Creek
Outcrop of Bed "A"
Helm Opening Bed "A"
Birch Creek
Outcrop of Bed "B"
Jackson Opening Bed "A"
W. & N. B. R. R.
To Williamsport
Outcrop of Bed "A"
Outcrop of Bed "A"
Outcrop of Bed "B"
Out Crop of Bed "B"
BERNICE
Mine No. 5
Mine No. 4
Mine No. 3
Mine No. 2
Mine No. 1
Mine No. 7
Mine No. 6
Obert's Creek
Pigeon Creek
Myler Opening Bed "A"
Outcrop of Bed "A"
Outcrop of Bed "B"
Outcrop of Bed "A"
Outcrop of Bed "A"
Outcrop of Bed "A"
Outcrop of Bed "A"
B. & S. R. R. AND TURNPIKE
LOYALSOCK
L. & T. R. R.
LOPEZ
Big Loyalsock Creek
Big Loyalsock Creek
Engd. by Am. Bank Note Co. N.Y.

about the smaller coal-basins in Wyoming county, east of the Bernice, indicates that the coal-beds of the Bernice basin are *in*, and not *over*, the Pottsville conglomerate. Mr. Ashburner says in his report:*

"There seems to be but little doubt that the Bernice coal-beds are the true geological representatives of the Lykens Valley coal-beds, which are found in the Pottsville conglomerate, No. XII., generally throughout the anthracite-region, but which are most favorably known from their occurrence in the Lykens Valley district, at the western end of the Southern anthracite field, where they are found with their maximum thickness and purity and have been extensively mined."

These coals from the Lykens Valley district are very similar in character and composition to the Bernice coals. The chief dissimilarity is in the color of the ash; the Lykens Valley ash being red and the Bernice white.

The cover over the coal is generally from 20 to 90 feet thick, but varies with the dip of the beds and the topography. The geological structure of the basin is comparatively uninteresting. Like all anthracite-basins, it conforms, in the main, to the canoe-shape. It underlies an area probably 1500 feet in average width, by nearly 4 miles in length, the longer axis lying nearly due east and west. The beds dip from 8 to 15 feet from the outcrop northward and southward to this axis, but contain many local "rolls" and "sumps," from which there is no water-level outlet, and which become very troublesome in mining. East and west the whole basin gradually dips from 30 to 70 feet towards its geographical center.

The two coal-beds included in the basin are locally known as the "A" and "B" beds. The lowest, "A," having nowhere been found to exceed from 10 to 20 inches in thickness, is without economic value. Bed "B," from 40 to 65 feet above "A," has been extensively mined during the past twenty years. It consists of three splits, known as the "Bottom," "Middle" or "Fourteen-inch," and "Top" coals or splits, the average thickness of each being as follows:

	Feet.	Inches.	Feet	Inches.
Bottom coal, ...	4	6	to 5	6
Middle or 14-inch coal, ...	1		to 2	
Top coal, ...	3		to 3	6

At the ends of the basin these three splits are separated by only a few inches of bony coal or slate, which parts easily, thus practically making one seam from 8 to 10 feet thick; but from the ends towards

* *Annual Report*, Geol. Survey, 1885, p. 471.

the middle of the basin they diverge, the top split running nearly level, while the fourteen-inch and bottom splits rapidly sink, so that, at the center of the basin, the interval between the top and fourteen-inch splits is from 45 to 60 feet, and that between the fourteen-inch and bottom splits from 15 to 25 feet.

The character of the Bernice coal is peculiar. It is an anthracite having, as it comes from the mines, a distinctly bituminous fracture. When loaded on cars, however, after preparation for market by crushing and screening, it looks very much like the ordinary run of the softer anthracites, and the bituminous fracture is more difficult to detect. So much does the coal, when in the form of "run of mine" (*i.e.*, unprepared), resemble ordinary bituminous coal, that former operators shipped it to market without any preliminary screening or sorting of the pieces, in which form it made a very unsatisfactory and undesirable fuel; and it was not until a suitable "breaker" was erected, and the coal was prepared in the usual commercial sizes, that it found anything like a ready sale.

The following table contains analyses made by different chemists and covering the period from 1871 to 1888:

Analyses of Coal from Bed "B" as Shipped to Market.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Moisture	.....	.....	.....	.....	0.96	1.97	1.295	1.81
Volatile matter	3.56	8.56	7.01	7.58	5.75	7.62	8.100	6.56
Fixed carbon	89.39	82.52	85.93	84.85	83.95	87.14	83.344	85.73
Sulphur	1.04	....	.....	.....	.....	....	1.031	.....
Ash	6.01	8.92	7.06	7.57	9.34	3.27	6.230	5 90
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.000	100.00
Sulphur	.....	.....	.....	.....	0.339	0.24	.....	.....
Phosphorus	.....	....	.....	.....	Trace	.....	.....	.....

Analysts: I., S. Dana Hayes, November 22d, 1871; II., Prof. F. A. Genth, December 8th, 1881; III., Dr. Chas. M. Cresson, November 2d, 1882; IV., Dr. Chas. M. Cresson, November 2d, 1882; V., Prof. Persifor Frazer, July 30th, 1883; VI., Salom and Westesson, November 18th, 1885; VII., Andrew McCreath (for Geo-

logical Survey of Pa., from specimens collected by Mr. Platt in 1885; see *Annual Report*, 1885, p. 477); VIII., Thomas Hayes, Toronto, February 10th, 1888.

In the first four analyses the amount of moisture is contained in the estimates of volatile matter. For purposes of comparison I have estimated this at 1.51 per cent., the average for the last four analyses, and have thus obtained the following carbon-ratios (ratios of volatile matter to fixed carbon):

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Carbon-ratio, 1:.....	43.6	11.7	15.6	13.9	14 6	11.4	10.3	13.08

According to Dr. Persifor Frazer's "Classification of Coals,"* first published in a paper presented to the Institute, and now generally accepted, five of these ratios would place Bernice coal among the anthracites, being over 1 : 12. The other three, being less than 1 : 12 would place it among the semi-anthracites; but the dividing line is very indistinct. According to a trade-classification, Bernice coal is a true anthracite, many of the Shamokin and Lykens Valley coals having a lower carbon-ratio.†

Analysis No. VIII. was made under peculiar circumstances. For many years the State Line and Sullivan Railroad Company, a miner and shipper of Bernice coal, had been shipping coal to Canadian points, and paying duty on it as anthracite according to the tariff law of the Dominion. About a year ago, the import-duty on anthracite was removed, while that on bituminous coal was retained; and thereupon the Dominion customs officials classed this coal as bituminous, and therefore dutiable. The duty was paid, but under protest, and letters and affidavits from Prof. Lesley, State Geologist of Pennsylvania, and others, were forwarded to the proper authorities, explaining

* *Trans.*, vi., 434.

† Mr. Ashburner, in a paper read before the Engineers' Club of Philadelphia in 1884, says of the Bernice coal: "This coal would be called a semi-anthracite on account of the percentages of volatile matter and fixed carbon contained. The percentage of volatile matter is not as great, however, as that contained in many of the coals from the Shamokin and Lykens Valley districts, with which the Loyalsock coal favorably compares. The Shamokin and Lykens Valley coals are called anthracites by the trade, and on this basis the trade must consider Loyalsock or Bernice coal an anthracite." But it should be here understood that Mr. Ashburner based his opinion on analysis No. VII., made for the Geological Survey, in which the carbon-ratio is considerably lower than in any of the others.

the true character of the coal. Since these did not seem to be sufficient to effect a reversal of the ruling, a sample of coal was finally taken from the yards of one of the consignees in Toronto, and analysis No. VIII. in the above table was made of it by Mr. Thomas Hayes of the Ontario School of Chemistry and Pharmacy. According to this analysis, the coal was entitled to rank as a true anthracite; and it was thereafter allowed to enter Canada free of duty.

Bernice coal ignites easily, burns with a very pale blue flame, and has the reputation of holding fire under conditions which would cause most other anthracites to "go out." It never clinkers, even under the strongest draft, but burns totally to a fine, impalpable ash. These characteristics make it a desirable fuel for domestic uses and for raising steam. In the ordinary locomotive fire-box its behavior is not satisfactory; the sharp exhaust "works" the fire, so that the coal breaks into small pieces and is either shaken through the grate-bars or blown out of the stack by the exhaust. But in a roomy fire-box, with large exhaust-nozzles, such as those used on "dirt-burners," there is no reason why it should not give satisfaction. For many years before the railroad of the State Line and Sullivan Company was leased by the Lehigh Valley Railroad the coal was burned with excellent results in the locomotives of the former company, which were adapted for its use.

The only mining operations in the Bernice basin are those of the State Line and Sullivan Railroad Company, which practically owns the entire area underlain by workable coal-beds. These operations are at present confined to the western end of the basin at Bernice, where seven drifts have been opened, only two of which are now producing coal. Owing to the slight dip North and South to the center-line of the basin, haulage was performed, until last summer, by mules; but recently, a new drift having been opened, a 9- by 14-inch mine-locomotive was introduced, to move the coal from a main turnout, 1500 feet inside, to the breaker. This locomotive, although only worked to half its capacity, has replaced eighteen mules and five drivers. Mining is done by the usual pillar-and-breast system; but the frequency of "rolls" or "sumps," from which there is no water-level outlet, renders a regular disposition of the mine-workings impossible. Where the interval between the top and bottom splits is not more than 8 feet, the top-coal is mined at the same time as the bottom; or the rooms are driven up in the bottom-coal and the top-coal is worked back, when the robbing of pillars takes place. But where this interval exceeds 8 feet, distinct mining operations are carried on

in each split, "break-throughs" or "cross-cuts" being kept as near as possible to the main working-faces in the top-coal by blasting down the roof in a bottom-coal room, grading it off, and laying a road upon it. In these cross-cuts either mules or self-acting inclines are used to move the coal. Drainage is either natural or is effected by syphons or pumps. Two years ago, a shaft was sunk 80 feet to the coal, near the center of the basin, and at its deepest point, as nearly as this could be determined, and was connected with the main workings by a long gangway in the coal. In this shaft a Davidson pump, with a six-inch discharge, takes care of all the water from the mines.

Former operators of the mines either neglected to mine the top-coal altogether, or failed to keep its workings in advance of those in the bottom; and, as a consequence, considerable difficulty was experienced in maintaining the top-coal haulage-roads, since the bottom-coal pillars, in settling, caused heavy falls of rock, thereby disturbing the floor of the top-coal. The present management has made special efforts to advance the upper workings beyond those in the bottom-coal; and not until this course was pursued was the difficulty of working removed.

Bernice coal is prepared for market in the usual smaller sizes, grate, egg, stove, chestnut, pea and buckwheat, there being no demand for it as lump or steamboat. The method of preparing the coal coincides, in the main, with that followed in the other anthracite districts, that is, it is a combination of crushing and screening, but the Bernice breaker has some peculiar features, the most important being the form of the crushing apparatus. Instead of passing between toothed or fluted rolls, as is the general custom, the coal is dumped from the mine-car (over the "mud-" screen bars) upon a large horizontal, perforated cast-iron table. This table, revolved by suitable gearing, carries the coal beneath the crusher proper, which consists of a cast-iron head, in which are set, by bolts, 32 steel teeth, and which is worked vertically up and down by means of eccentrics, somewhat after the manner of a stamp in a stamp-mill. The coal carried around by the table is struck by the crusher-teeth, broken into small pieces, and so carried off. The shape of the teeth, relative speed of the table to the number of strokes of the crusher, and distance of the teeth from the table, are conditions determining the size of coal obtained, and can be regulated at will. This machinery appears, at first sight, to be crude; but extensive tests show that it operates successfully on coals like

the Bernice, which require, by reason of their softness and structure, peculiar treatment in preparation. From the crusher the coal goes directly to screens of the ordinary revolving type, by which it is separated into the various commercial sizes.

At present, the only railroad tapping the Bernice field is the State Line and Sullivan Railroad, owned by the company of that name, but operated under lease since 1884 by the Lehigh Valley Railroad. This road runs from the mines at Bernice, thirty miles to Towanda, on the main line of the Lehigh Valley Railroad. Next year, however, there will be extensive developments in the railroads of Sullivan county. The Lehigh Valley Railroad has already extended the State Line and Sullivan road some fifteen miles southeast of Bernice, the plan being to connect it with the Harvey's Lake branch of that road coming up toward Bernice from Coxton. The Bloomsburg and Sullivan Railroad has already completed a line from Bloomsburg to Jamison City, at the foot of North Mountain, and proposes to complete the road to Bernice within the next eighteen months. The Williamsport and North Branch Railroad, which is already built from Hall's Station on the North branch of the Susquehanna to Nordmont, only fourteen miles from Bernice, will complete the line to Bernice during the coming season. And, lastly, the Williamsport and Binghampton Railroad, projected to connect the two cities of Williamsport and Binghampton, and now under contract for construction, will tap the State Line and Sullivan Railroad near Dushore, only four miles from Bernice. Possessing the extensive mines of the State Line and Sullivan Railroad Company, and thus becoming a center for five railroads, Bernice should soon be a thriving town.

The total product of coal from the Bernice field, from 1871 (the earliest year for which records are obtainable, though the mines were opened in 1869) to 1886 was 1,197,236 tons (2240 pounds). In 1887 the product amounted, for the year, to 96,000 tons and in 1888 to 105,107 tons. Bernice coal finds its best market throughout New York State and the West; and in Baltimore and Washington, where the soft free-burning anthracite coals are used, it finds a ready sale.

The small coal-bed "A," occurring from 40 to 65 feet below "B," while of no economical importance, possesses considerable geological interest by reason of the fact, indicated by all the analyses now at hand, that it changes in character from anthracite to bituminous coal within less than a mile and a half. This bed has only been opened in three places: the "Mylert opening," near the extreme

western end of the field; the "Jackson opening," one and a half miles east of the Mylert opening and near the town of Bernice; and the "Heiss opening," about five miles east of the Jackson opening, and near the eastern end of the field.

In 1886 I reopened the Mylert opening, drifted in on the coal, and took specimens which, being analyzed by Messrs. Salom and Westesson, of Philadelphia, gave the following:

	Per cent.
Moisture,	1.43
Volatile matter,	10.17
Fixed carbon,	85.72
Ash,	2.68
	100.00
Sulphur,	0.12
Carbon-ratio,	1:8.4

The Jackson opening was sampled by Mr. Platt in 1877, and again by Mr. Ashburner in 1884; and the samples being analyzed by Mr. McCreath gave:

	1877. Per cent.	1884. Per cent.
Moisture,	4.130	3 670
Volatile matter,	15.270	15.420
Fixed carbon,	67.362	71.341
Sulphur,	.523	.594
Ash,	12.715	8.975
	100.000	100.000
Carbon ratio,	1: 4.4	1: 4.6

The Heiss opening has never been sampled for analysis. I have several times visited it for that purpose; but sufficiently good samples of coal could not be obtained, except by the expenditure of considerable time and money.

Upon examining the foregoing analyses it will be seen that the Mylert coal shows a semi-anthracitic composition, or, by trade classification, that of an anthracite; while the coal from the Jackson opening appears to be true bituminous coal. This change takes place within one and a half miles. It would be interesting to know just where the merging line from one to the other is located. The character of the coal at either one or the other of these openings may be only local and cover but a small area. It would be interesting to geologists also to have the character of this bed clearly defined, as it might lead to the final settlement of some vexed questions as to

the mode of formation of anthracite coal. If this bed "A" is a true bituminous bed, occurring as it does 40 to 65 feet *below* a true anthracite bed ("B"), the theory of the formation of anthracite by the action of heat on bituminous coal could scarcely be applied to this case; for it is not easy to conceive of any heat, apart from an eruptive overflow, affecting a certain coal-bed over a considerable area, and not affecting another bed only 40 feet below it.

THE DISTRIBUTION OF PHOSPHORUS IN THE LUDINGTON MINE, IRON MOUNTAIN, MICHIGAN: A STUDY IN ISOCHEMIC LINES.

BY DAVID H. BROWNE, ANN ARBOR, MICH.

(New York Meeting, February, 1889.)

ONE of the most difficult problems in the chemistry of iron-ore, and one, the solution of which, so far as I am aware, has never been attempted, is the distribution, throughout a given vein, of Bessemer ore, and its relation to the formation of the deposit. In those hematite-mines in which both Bessemer and non-Bessemer ore occur, the sorting of the ore as it lies in the deposit becomes a problem of much economic as well as scientific interest. It would seem, from a superficial examination, or, indeed, from any examination not conducted for this especial purpose, as if high- and low-phosphorus ores were mixed in inexplicable confusion; and a mining-chemist is very apt to fall into a system of adventitious analyses, taking first-class ore wherever he can find it, and overlooking its relation to the formation and position of the vein. I hope that a few notes which I present on this subject may be found worthy of consideration, as throwing a new light upon this obscure topic.

During the last three years, while acting as chemist of the Lumberman's Mining Co., I have made some 3000 analyses of iron-ore from the Ludington mine at Iron Mountain, Michigan. These analyses were necessary in order to separate Bessemer and non-Bessemer ore which occurred intermixed in the deposit. During the last year, I attempted in several ways to find some reason or method in the distribution of phosphorus, and have finally become cognizant of the arrangement herein outlined. I have been obliged to confine my attention to our new mine, and, in that, to that portion wherein

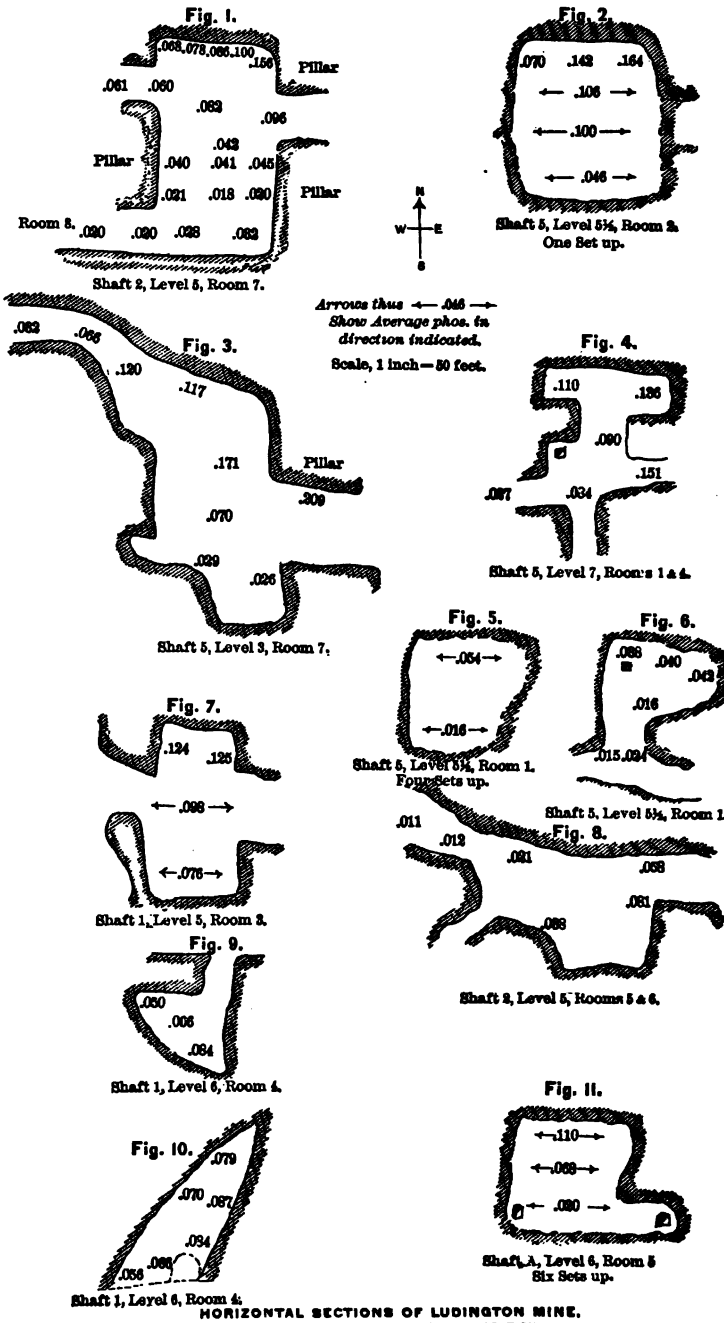
the commercial quality of the ore was such as to demand for it careful sampling and separation. The analyses and conclusions drawn therefrom are given merely as facts found to exist, and I do not claim that such sequence as I have noticed will obtain in all or every case. I simply state what results I have obtained in an instance falling under my own observation, and I give some conclusions toward which the data seem to lead.

The Ludington mine, like most on the Menominee range, consists of several lenticular deposits of soft blue hematite. These deposits are contained between clay and soap slates, which conform with the Huronian strata represented in the district. The main deposit is about 700 feet in length; striking N. 75° W., pitching about 45° W., and dipping from 70° to 80° N. The ore is a very rich, soft, friable, bluish-black hematite, occurring in thin laminæ, which cleave from each other very readily in the direction of the strike of the deposit. These layers alternate in places with layers of calcium-magnesium carbonate. The ore shows by analysis from 65 to 68 per cent. iron, 1 to 4 per cent. silica, and from 0.005 to 0.200 per cent. phosphorus. It is separated into Bessemer and non-Bessemer; about one-half falling below 0.035 phosphorus, and the rest averaging, perhaps, 0.075.

At first sight the ore, upon analysis, seems to have no regularity or method in its distribution of phosphorus. A room, as stoped out, will change from Bessemer to non-Bessemer, or *vice versa*, in a way at first totally inexplicable.

The fact that phosphorus exists as calcium phosphate led me to infer that some proportion between the percentage of lime and phosphorus might be found to exist; but such inference was not verified in practice. An ore containing 2 per cent. of lime may contain almost no phosphorus, or may run high above Bessemer limits. Nor does any proportion exist between the amount of silica or of iron, and the phosphorus content. I have seen phosphorus vary as much in jasper as in any iron-ore; and similarly a lean ore may run high or low in its percentage of phosphorus. The only difference I could find between Bessemer and non-Bessemer ores was this: As a rule, a soft blue hematite, high in phosphorus, has a brighter and more specular appearance than a low-phosphorus ore of the same value in iron. This distinction, slight as it is, does not always hold good, and the separation of such ores must be guided solely by chemical analyses.

The fact that the bright ore is generally high in phosphorus, and



HORIZONTAL SECTIONS OF LUDINGTON MINE.
EXCEPT FIG. 10. WHICH IS A VERTICAL SECTION.

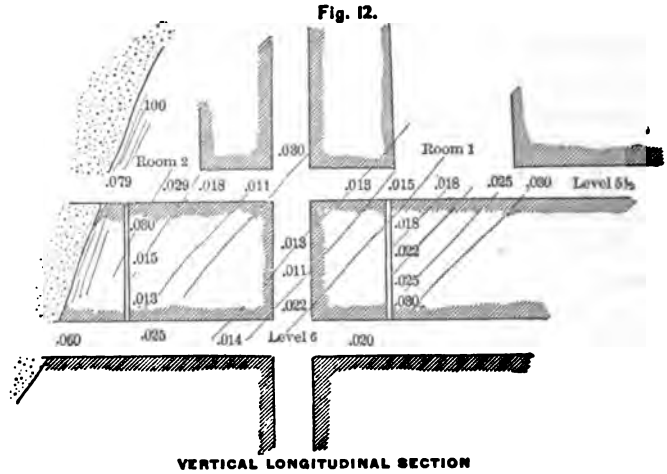
that such ore is almost always found near the hanging-wall, led me to search for some regularity of phosphorus-distribution dependent upon the position of the ore. After making an analysis of the ore from any room, drift, or winze, I marked the percentage of phosphorus in a map of that portion of the mine. Having thus obtained a chemical map of each room, I noticed in each a certain regularity which seemed to me to throw considerable light both upon this problem of phosphorus-distribution, and upon the vexed question of the method of formation of soft blue hematite deposits.

In order to give a clear idea of this relation, I must first state a few facts with regard to the physical features of the deposit. As previously observed, the so-called "veins" of the Ludington mine stand nearly vertical, dipping north and pitching west. A horizontal cross-section of the ore-body shows it to form an elongated lens, about 65 feet thick at the center, and tapering to an acute point at both ends. (See Fig. 16.) A vertical cross-section shows the dip to the north, and also the fact that the hanging-wall is more curved than the foot. A cross-section of the Chapin mine, which possesses the same physical features as the Ludington, shows this very plainly. (See Fig. 17.)

A horizontal cross-section of a small vein shows that the hanging-wall curves toward the foot. On large veins, the strata have been subjected to so much lateral flexure that this curvature is not clearly seen. On small veins it is unmistakable. (Fig. 18.) I must here observe that in the greater number of mines on the Menominee range the dip is to the south; and hence what is called the hanging-wall in the Chapin and Ludington mines answers in other mines to the foot-wall. If we attempt to make an ideal vertical longitudinal section of the ore-deposit, it seems to have the shape outlined in Fig. 19. A study of the eastern and western limits of the Chapin mine seems to verify this idea. Fig. 13 gives a vertical longitudinal section of a small vein in the Ludington mine, in which this shape is very clearly seen. The ore will now be understood to lie in the form of lenticular deposits, dipping north and pitching west.

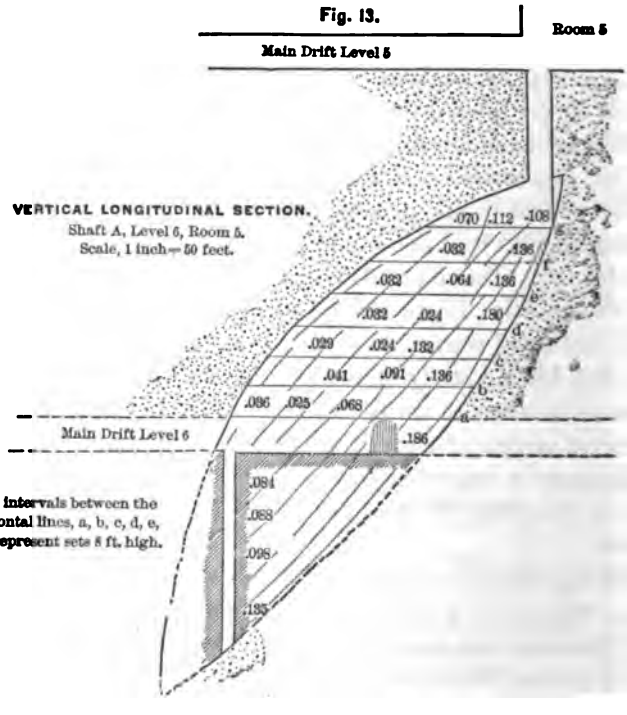
With regard to the content of phosphorus, the first thing noticeable was that if a room, in stoping up, changed from non-Bessemer to Bessemer ore, such change was liable to occur first at the foot-wall side of the room. In making maps of those rooms in which change occurred, it was also noticeable that the ore at the eastern end of the rooms was higher in phosphorus than that at the western end.

Fig. 12.



VERTICAL LONGITUDINAL SECTION
Levels 5½ & 6, Rooms 1 & 2.
Scale, 1 inch = 50 feet.

Fig. 13.



(See Fig. 2.) The most typical room was Shaft 2, Level 5, Room 7, which is outlined in Fig. 1. This room was, if I remember rightly, about 3½ sets from east to west, and 6 sets from foot to hanging. (A set is a space 8 by 8 by 8 feet, outlined by the timbers used to support the back.) From the ground-plan of this room, it will be noticed that the ore showed a marked regularity of formation. Follow the course of the hanging-wall, and trace the increase of phosphorus from 0.068 at the west hanging-wall set to 0.078, 0.086, 0.100, and finally 0.156 on the east hanging-wall set. In the same manner, though less strikingly, the ore on the foot-wall shows increase from 0.020 in Room 8 to 0.032 on the east foot-wall set in Room 7. Besides this regularity from west to east there is a corresponding increase from foot-wall to hanging. Notice the gradual change from 0.032 and 0.028, on the foot-wall, to 0.045 and 0.040 in the middle, and 0.156 and 0.068 on the hanging. The streak of low-phosphorus ore, 0.021, 0.018, 0.020, I will speak of again. On inspection, Figs. 1 to 11 will also show the same peculiarity; and a large number of average analyses corroborate the conclusion that, as a rule, in this mine, the ore increases in percentage of phosphorus from foot-wall to hanging, and, generally speaking, from west to east. It frequently happens, however, that a streak of high- or low-phosphorus ore crosses a room from west to east, as in Figs. 1, 14 and 15. This seems to be due to the fact that one or more individual layers of ore were originally either very high or very low in phosphorus, and this individuality has not been obscured by subsequent changes. Moreover, an irregularity is often noticed in the increase from west to east. This tendency is not nearly so uniform, or so well marked, as is the increase from foot-wall to hanging-wall. Nor is this to be wondered at; for, since the layers of ore present smooth surfaces in the direction of foot and hanging-walls, an average of the ore along a series of sets on the foot-wall will represent, roughly, analyses of at most a very few layers of ore. In this way it will be seen why analyses taken from foot to hanging show more regularity of composition than do analyses taken east and west. In driving a drift or stoping a room, on the other hand, where analyses are taken in an eastern and western direction, averages are taken of a large number of separate deposits, and as these are much flexed and broken, the analyses show little method or correspondence. Supposing each layer of ore has a thickness of half an inch (which is much above the average), there will present themselves for analysis, in the breast of a drift 8 feet wide, the edges of no less than 192 layers; and, in consequence, more con-

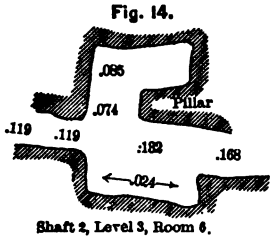
fusion is liable, and does occur, in analyses taken east and west than in those taken north and south.

Having obtained thus a general idea of how the lines of phosphorus tend in two directions, the next question naturally is, what would be the lines of equal phosphorus-content in any individual layer of ore. These, for want of a better term, I have herein been obliged to term "isochemic" lines. It is evident that mere analyses of the ore in the breast of a drift or in the bottom of a winze would not give any clue to the isochemic lines of a particular stratum of ore. They would simply show the average of some hundred separate strata. It is also evident that no analysis would represent accurately the character of an individual layer, unless this layer, in no case over half an inch, and rarely over one-quarter inch in thickness, could be followed by chemical analyses along drifts, and up stopes, and down winzes and shafts, for a distance in some way proportionate to the extent covered by the ore-deposit, of the thickness of which it forms a very small part. This would be, and for me was, practically impossible. For analyses, to be of commercial value, must show, not the amount of constituents in any particular stratum of a deposit, but an average of that amount of ore which a gang of men, working under contract, can take out of a given room before another analysis be made. For this reason I have been obliged to confine myself to analyses which represent averages of perhaps 200 or more layers, and from these analyses I have attempted to outline the probable direction of isochemic lines in separate strata. It is plain that, if a single layer of ore shall have its percentage of phosphorus in some way modified by its manner of deposition, every other layer of ore, subjected to similar conditions, will be in similar manner modified; and, in consequence, analyses representing averages of a large number of strata, will show the characteristics common to each individual stratum.

In sinking a winze in Shaft 5, Level 5½, Room 1, the following facts were noticed: The drift running east from the winze showed ore running from 0.013 phosphorus to 0.030; the winze as sunk passed through ore running from 0.015 up to 0.030; lines drawn from the point in the drift where a certain amount of phosphorus was noted to a corresponding point in the winze, showed an angle of about 45°, which corresponded with the pitch of the ore. (Fig. 12.)

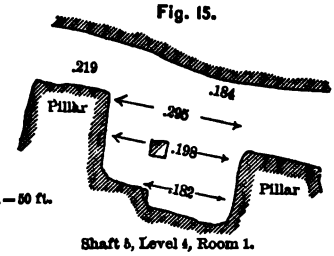
Again, in sinking the winze in Shaft 5, Level 5½, Room 2, a decrease in amount of phosphorus was noted, which was paralleled by the decrease in the amount of phosphorus in the entry to the east. Similarly, in sinking Shaft 5, and in sinking winzes from Rooms 2

Fig. 14.



Shaft 2, Level 3, Room 6.

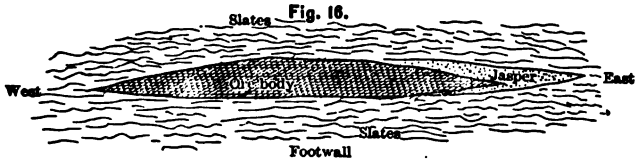
Fig. 15.



Scale, 1 inch = 50 ft.

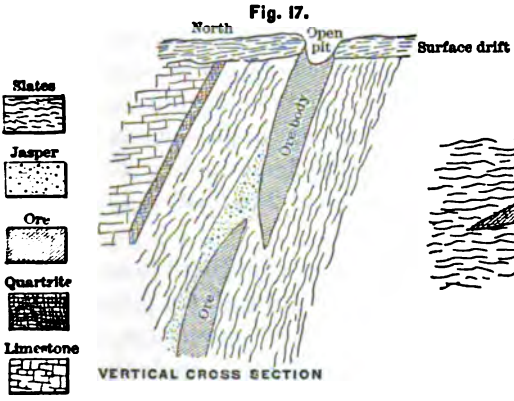
Shaft 5, Level 4, Room 1.

Fig. 16.



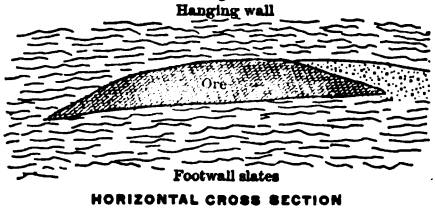
HORIZONTAL SECTION

Fig. 17.



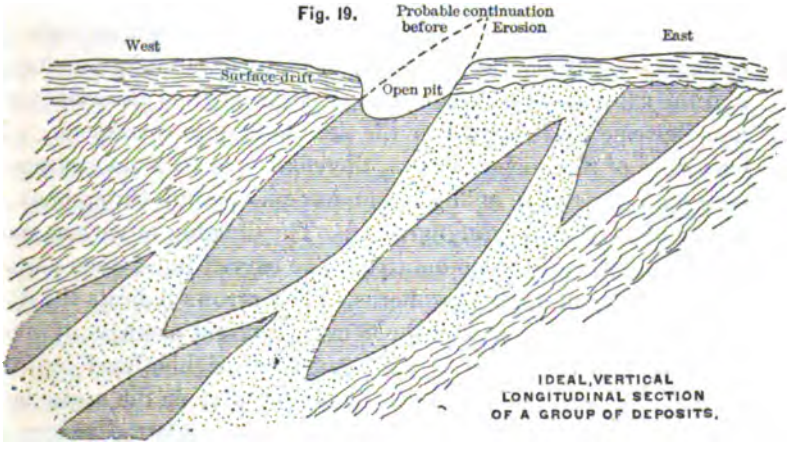
VERTICAL CROSS SECTION

Fig. 18.



HORIZONTAL CROSS SECTION

Fig. 19.



IDEAL, VERTICAL LONGITUDINAL SECTION OF A GROUP OF DEPOSITS.

and 3 on the seventh level, the continuance of isochemic lines, with a pitch of 40° to 50°, was noticed. (Fig. 21.)

If now we take up a small vein, as that composing Rooms 4 and 5 of Shaft A, Level 6, and attempt to outline the isochemic lines in the plane of the winzes, the regularity is at once patent. On the level of the entry the first-class ore was confined to the west set, and one-half of the next set east. As the room was stoped up, the first-class ore seemed to widen and gain somewhat toward the east. (Fig. 13.) Analyses taken of a small and very characteristic vein, forming Room 4 of Shaft 1, Level 6, showed also the tendency of the phosphorus to gain towards the upper part of the deposit and to maintain uniformity in the direction of the dip of the deposit. (Fig. 10.)

In attempting to draw up a vertical longitudinal section of the western end of the mine the principal difficulty lay in reducing so many analyses to the same plane. In consequence of the impracticability of attempting to represent every analysis taken, and its relation to others in the same vein, I have been obliged to select those analyses which represent averages. The figures entered in the drawing represent, therefore, the average percentage of phosphorus in the ore in that particular place covered by the figures. Fig. 20 gives the detail of various averages in Rooms 1 and 2 of Shaft 5, Levels 5 and 6, in the plane of the shaft.

While the ore left in the pillars has not been subjected to analysis, I have, for the sake of convenience, drawn through the pillars the isochemic lines indicated by the percentage of phosphorus in the rooms which they support. Fig. 23 is an attempt to apply the same system to the entire course of Shaft 5. By actual measurement the distances along various levels, through which certain average percentages of phosphorus obtained, have been carefully ascertained, and the exact point at which change occurred from Bessemer ore to non-Bessemer, was located in the section-map to correspond. The curvature of the isochemic lines, therefore, is in accurate correspondence with the course of high- and low-phosphorus throughout the western end of the Ludington mine. The drawing of various rooms and pillars is in this map omitted. The curved lines, when close together, represent high phosphorus. The arrows indicate the direction of low-phosphorus, and the figures represent averages of several hundred analyses, taken in the immediate neighborhood indicated thereby. From the map it will be noticed that on the upper levels the greater portion of the ore is non-Bessemer. On the western end a small streak of Bessemer ore follows the shaft, gaining to the west

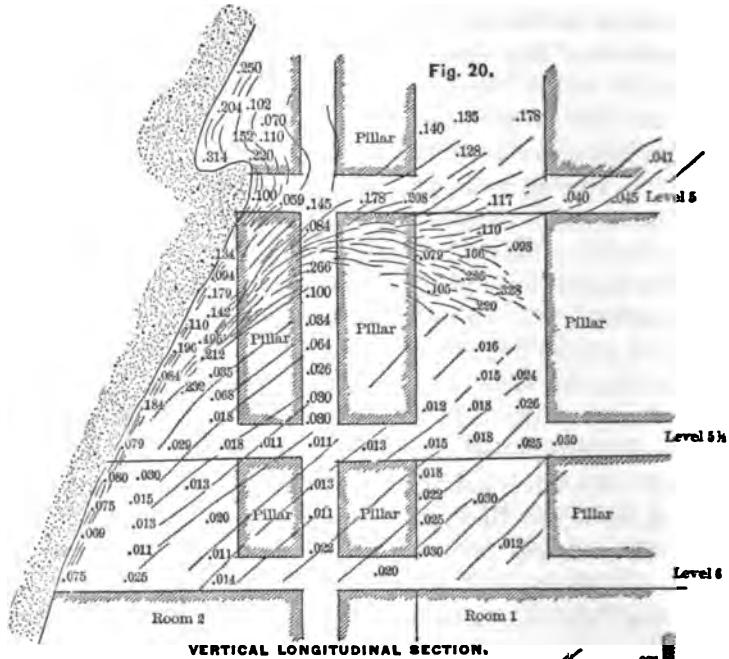
until it reaches the fifth level, where it flexes east and joins the main current of first-class ore that flows upward and eastward from the lower end of Shaft 5. The non-Bessemer ore follows the western boundary of rock, and seems to accumulate in shoals outlined by projections or intrusions of jasper.

In stoping up Room 1 of Shaft 5, Level 7, through ore gradually increasing in phosphorus, a thin seam of rock was encountered. As analysis of the drift 100 feet overhead had shown low phosphorus, I concluded that the ore found over this rock would be of first-class quality. The winzes in Room 1 had not at this time been sunk, nor had any ore from above the rock been taken for analysis. The only indication offered of its probable quality was that afforded by the passage of the drift 100 feet above, and of the shaft 50 feet to the west through first-class ore. Analyses had been made of two sides of a triangle; and, upon the supposed consistency and continuance of isochemic lines, prediction was made that the ore found above this rock was first-class. I was at this point called to New York on business; and, before leaving, left word with the mining captain that any ore found above this rock should be sent up for Bessemer ore. On returning to the mine some three weeks later, I found that the rock in this room had been pierced and about 200 tons of ore from overhead sent up and dumped on the first-class stock-pile. Analysis of this ore showed it to be from 0.011 to 0.027 phosphorus, which will be seen to agree with other analyses along the isochemic line. (Fig. 21.)

Below this seventh level, at the western part of the vein, the intrusion of rock seems to have caused an inflow of high-phosphorus ore. It appears as if this rock had formed shoal-water on its lower side, and into this shallow the lighter particles of calcium phosphate had drifted. In a large number of instances I have noticed this tendency of rock, occurring in vein-matter, to alter the percentage of phosphorus in the adjoining ore. In fact, I do not remember any instance in which a horse of jasper did not in some way alter the proportion of phosphorus in the ore penetrated by it. The statement that high phosphorus follows rock is one which is corroborated by every one familiar with the mine under consideration.

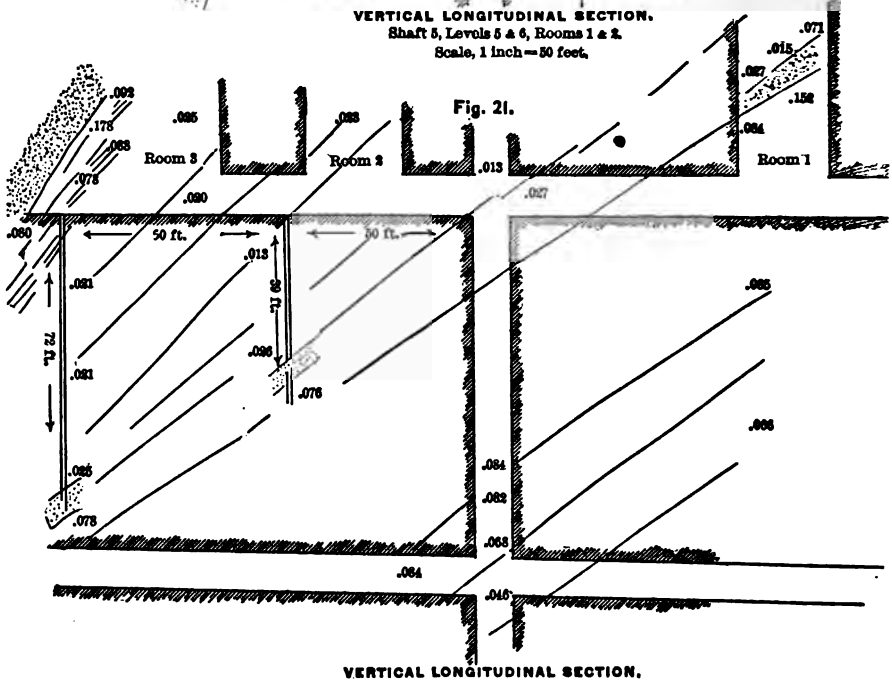
Another fact I must state is this: On the upper levels of Shaft 5 almost all the non-Bessemer ore mined was exceedingly high in phosphorus. The ore found on the lower levels shows a greater tendency to uniformity; the difference between ore below and above Bessemer limit being less striking. A glance at the sketch-map,

Fig. 20.



VERTICAL LONGITUDINAL SECTION.
Shaft 5, Levels 5 & 6, Rooms 1 & 2.
Scale, 1 inch = 50 feet.

Fig. 21.



VERTICAL LONGITUDINAL SECTION.
Level 7, Rooms 1, 2, & 3;
Showing depth of winzes.

Fig. 23, will show this clearly. It was no uncommon occurrence on the third and fourth level, and in the western end of the open pit, to find streaks of ore running as high as 0.350 phosphorus. Now, such ore is very rarely met with. The average of non-Bessemer ore on the upper levels of Shaft 5 was somewhere near 0.150 phosphorus. The average of non-Bessemer at present drawn from Shaft 5 is probably 0.075 or 0.080. Again, on the upper levels, some streaks of low-phosphorus ore were found, but in narrow, irregular courses. On the lower levels the reverse of this is true, the first-class ore occupying the larger part of the deposit.

To state this in general terms: The tendency of phosphorus, on this vein, is to increase in percentage with the distance from the lower western extremity of the deposit. This is not true of the eastern end of the mine, in which the upper portion of the deposit is low-phosphorus. This low percentage seems due to the fact that a large horse of jasper, occurring near the middle of the vein, has thrown the current of Bessemer ore to the east; and below this comes the drift of high-phosphorus ore, met with on the lower levels of Shaft 5.

In endeavoring to correlate these isochemic lines with the physical phenomena of the deposit, the only theory which will, to my mind, furnish adequate explanation, is that of aqueous deposition. The easy longitudinal cleavage of the laminæ of ore, the curvature in small veins of the hanging-wall towards the foot, the numerous ripple-marks found on the hanging-wall and the hydrated muddy-looking ore next to the foot-wall, all seem to indicate that the ore was deposited from water in hollows of the exposed slates which now form the hanging-wall. Furthermore, since it is well understood that the almost uniform tendency of all deposits east of the Mississippi river is in a line from southwest to northeast, it is very probable that this deposit, as originally laid down, was no exception to the general rule. If we suppose that the ore was formed in hollows in the hanging-wall, and, after its formation, covered by the foot-wall slates; and that this bed has been tilted up from the north side and overturned, through an angle of 100° to 110°, it will be readily understood that the original trend of the deposit becomes approximately the complement of the present pitch of the ore. This supposition explains also the strike of N. 75° W., and the fact that what is now the hanging-wall seems to have been the original bed of deposit.

It is improbable that the tilting has been from the south side upwards through an angle of 70° to 80°, for if this had been the case

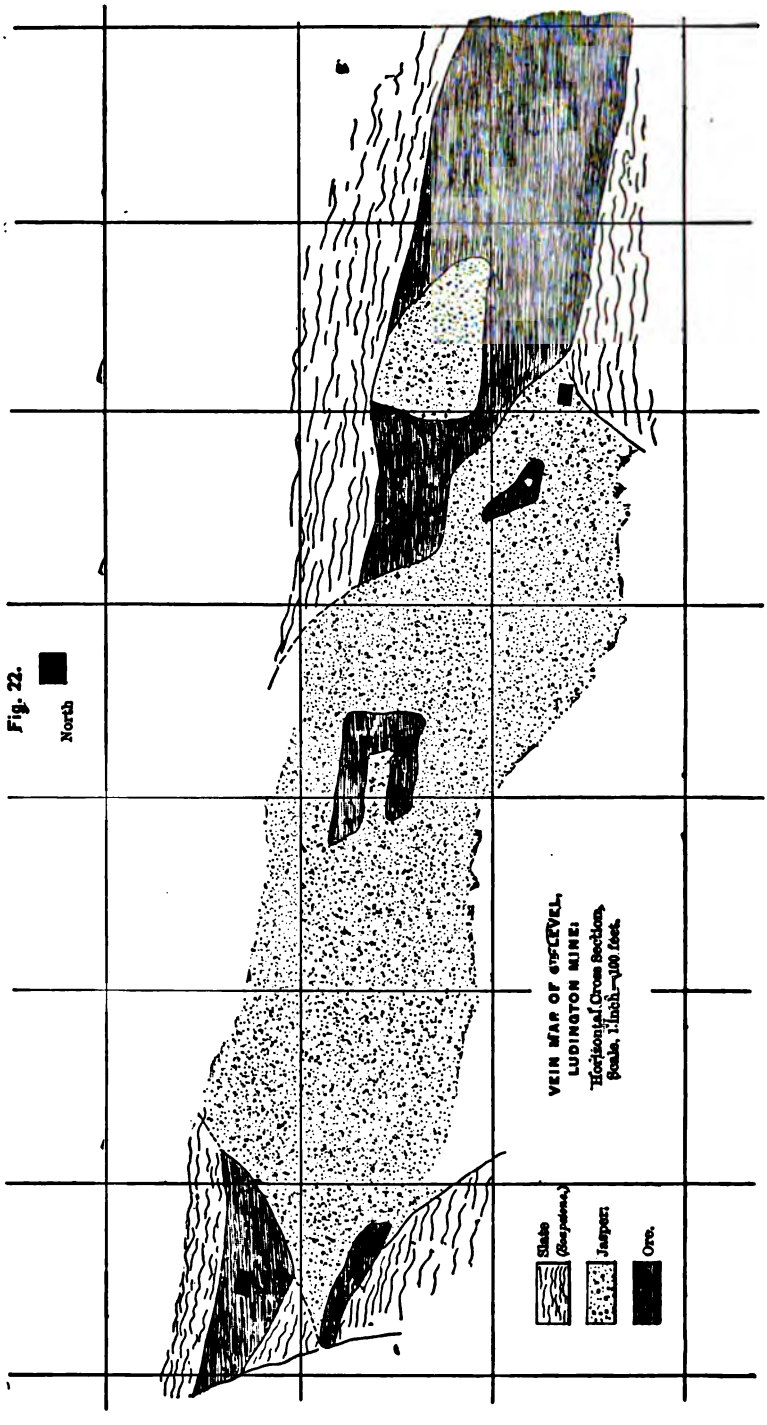
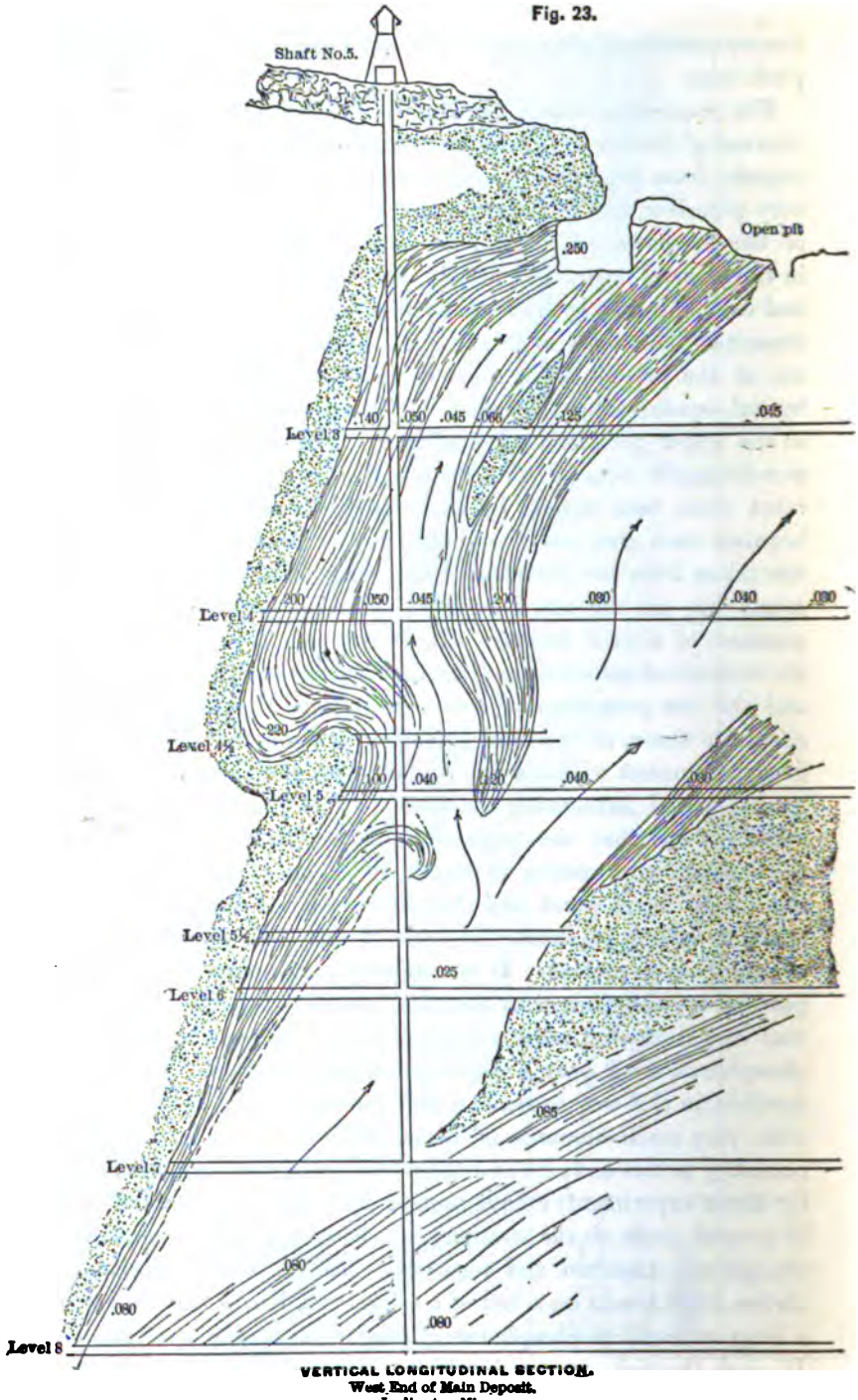

VEIN MAP OF 6TH LEVEL,
LUDINGTON MINE:
Horizontal Cross Section,
Scale, 1 Inch = 100 feet.
Slate (Soapstone.)
Jasper.
Ore.
North

Fig. 22.

the ore would now pitch east at the same angle as that of the present pitch west.

The suggestion made by Mr. R. D. Irving in the *American Journal of Science* for 1886, that the ore has been washed into these deposits from previously precipitated beds of carbonate, seems to me very plausible, and is borne out, in large measure, by the chemistry of the ore-body. I should, however, suggest this change: that in the original deposits, iron and lime were present, not as siderite and calcite, but as hydrous oxide of iron, and intermixed calcareous deposits. This finds analogy in the formation of beds of bog-iron ore at the present day. Such deposits of bog-ore, altered indeed by subsequent action, do exist in the Huronian strata represented in the upper peninsula of Michigan. Any one familiar with the non-Bessemer ores of the western end of the Menominee range must have been struck by the remarkable dissimilarity existing between such ores and those east of the Menominee river. The ore taken from the Nanaimo, Paint River, Iron River and other mines has all the non-laminated, massive, porous, yellowish appearance of altered bog-ore. With this also the high phosphorus, the intermixed calcareous matter, the frequent occurrence of graphite, and the low percentage of iron seem to agree. The abundance of graphitic slates, in the same geological horizon, and in some places in actual contact with the ore, suggests that the fucoids of the Huronian period have taken the same part in the deposition of these western beds that the vegetation of the Carboniferous Era, and the swamps and mosses of recent times, have played in the formation of black-band and bog-iron ores. From such beds as these, I incline to think, the iron of our soft hematite-ores has originally been carried. It is interesting to notice how this supposition is strengthened by chemical proofs. I have often noticed that very dilute hot acids will dissolve from an ore almost all of the phosphorus, with only a slight percentage of iron. Indeed, it is possible in this way to remove and estimate all of the phosphorus with very small amounts of dissolved iron. Nor is acid always necessary to this end; for in a large number of instances, verified also by direct experiment, I have noticed that ore, exposed for a period of several years to the weather, will have appreciable amounts of phosphorus dissolved and removed. Now, if water acidulated by carbon dioxide acts on a bed of bog-iron ore, it will carry therefrom a large amount of phosphorus in proportion to the iron removed. If such thermal water flow into a shallow valley or lake, the acid

Fig. 23.



VERTICAL LONGITUDINAL SECTION.
West End of Main Deposit.
Ludington Mine.
Scale. 1 inch=100 feet.

will be lost by evaporation and consequent precipitation of phosphorus and iron will take place.

The theory of hematite-deposits is then briefly as follows: From previously-deposited beds of bog-iron ore, by the action of acidulated water, iron, silica, lime and phosphorus were dissolved. The first solution contained a large amount of phosphorus in proportion to the amount of iron dissolved. On coming into depressions in the exposed slates, this acid solution, losing acid by evaporation, deposited iron as hydrated oxide, which carried down an amount of phosphorus proportional to the amount of iron precipitated. As the acid became still weaker, crystals of carbonate of lime and magnesia settled out, forming a stratum of carbonates. A second inflow of acid water would tend to dissolve part of the carbonates, and to precipitate a layer of hydrated oxide of iron, similar to the first, but lower in phosphorus.

In similar manner by successive inundations the depression became filled with alternating deposits of iron oxy-hydrate and calcium and magnesium carbonate, each layer being, as a rule, lower in phosphorus than that immediately preceding it. The carbonates being more soluble were more liable than the oxy-hydrate to removal by acid. Moreover, as both phosphate of lime and phosphate of iron are more soluble and of lower specific gravity than oxy-hydrate, the tendency of the water was to carry these phosphates toward the lower end of the ore-deposit, and to form deposits higher in phosphorus in the shoal-water alongside banks of previously precipitated silica, or in places where the evaporation progressed more rapidly than elsewhere. By reference to Fig. 23, it will be seen that those parts of the deposit in which the current seems to have been strong and rapid are lower in phosphorus than where the deposit is shallow and pinched out by jasper. After the deposit was completed, further action of the water would stir up the upper layers of ore and mix them with suspended sand or clay, while the iron and phosphorus were carried further along to be deposited in other depressions to the northeast. The jasper, which occurs as vein-matter, and in laminæ cleaving parallel with the ore, seems to have been produced, either by precipitation at the same time as the iron, or by subsequent action, eroding the beds of iron-ore thus formed and substituting silica for the iron oxide removed. A study of the vein-map of the sixth level of the Ludington mine shows that the jasper seems bedded in the ore; the deposits of jasper being in all cases widest toward the foot-wall. (See Fig. 22.)

The deposits of ore and jasper were, I conclude, covered by the foot-wall slates, and, after subjection to heat and pressure from superincumbent strata were, in some local upheaval, tipped up at the north and brought into their present position.

That subsequent action of percolating surface-water has wrought changes in the distribution of both iron and phosphorus, I do not doubt. In fact, the western end of the Ludington mine, protected by overhanging jasper, has shown far greater regularity of structure than the eastern end of the same deposit. In the former, the original lines have been preserved unchanged; in the latter, where the thin cover of drift allows free action of surface-water, the lines are much broken, and regularity of structure is hard to trace.

This theory of aqueous deposit will explain, as will no other, the marked continuance of isochemic lines and their peculiar curves, the regular decrease of phosphorus from hanging-wall to foot, the alternation of carbonates with oxide of iron, the ripple-marked hanging-wall, the uniform lamination of the ore, and the hydrated muddy look of the ore next to the foot-wall.

The conclusions given above are intended, not as general and applicable in all cases, but simply as an explanation of certain chemical phenomena, noticed in the Ludington mine. Whether the same tendency would be found to exist in other mines, I am not prepared to say; but the fact that apparent irregularity has been observed does not preclude the discovery of a law—in other words, an order—pervading it. I incline to think that a careful and systematic method of chemical research, applied to the soft-ore deposits of the Upper Peninsula of Michigan, would bring to light many interesting facts with regard to their manner of deposit, and would lead to a more thorough understanding of one of the most interesting and practical problems of economic geology.

In conclusion, I would acknowledge my indebtedness to Mr. Per Larsson, Engineer of the Chapin mine, to Mr. Chas. H. Snow, Engineer of the Ludington mine, and to Mr. E. Everett, of Ishpeming, for the maps from which the plates illustrating this article have been drawn.

NOTE ON THE INFLUENCE OF COLUMBITE UPON THE TIN-ASSAY.

BY FRANKLIN R. CARPENTER AND W. P. HEADDEN, RAPID CITY, DAKOTA.

(New York Meeting, February, 1889.)

Two notes have already appeared in the *Transactions* concerning the columbite or tantalite of the Black Hills tin-mines. In vol. xiii., page 232, Prof. Schaeffer speaks of the mineral as tantalite, and concludes by saying that its separation from the tinstone "is a matter which will, I believe, prove puzzling to the wisest metallurgist." He also observes that he is unable to find by analysis in his specimens the least trace of columbic acid. Prof. W. P. Blake, in the same volume, at page 696, says: "Even if it [tantalite or columbite] were present to a considerable amount, it would not injure the tin in smelting. It does not alloy with tin, but carries a small percentage of tin, which would probably be liberated in smelting." He calls the mineral columbite from the fact that a great number of specific-gravity tests invariably indicated that it should be so classed.

We have made chemical tests as well as specific-gravity tests upon many samples from many different parts of the Hills. Our analyses have always shown the presence of columbic acid, and the specific-gravity tests have always indicated columbite, as we have elsewhere shown.*

As Prof. Blake has already observed, it is not found in such close association with the tin as to prevent separation by a rough hand-sorting, but it is usually present with the stream-tin. As Prof. Schaeffer says, it cannot be separated by ordinary concentration; and while, as declared by Prof. Blake, its columbic acid may not unite with the tin in any form, yet its iron and manganese oxides may be reduced, and thus lower the quality of the tin produced, as is also the case when the smelting charge contains wolfram.

To determine the effect of columbite in the ordinary potassium cyanide assay, there were assayed at the Dakota School of Mines four samples containing different proportions of columbite. The results, published in the report of the school for 1888, seemed to indi-

* *Preliminary Report of the Dakota School of Mines upon the Geology, Mineral Resources, and Mills of the Black Hills of Dakota*, 1888. See also Dr. Carpenter's paper on the "Ore-Deposits of the Black Hills," at p. 570 of the present volume.

cate that the presence of columbite would not affect the accuracy of the assay. The subject has, however, been re-investigated by the present writers, who have arrived at a contrary conclusion.

The columbite used in our experiments showed upon analysis the following composition:

Ta_2O_5,	18.20
Nb_2O_5,	64.09
FeO,	11.21
MnO,	7.07
SnO_2,	0.10
CaO,	0.21
	100.88
Specific gravity,	5.89

The cassiterite used in the experiments was Dakota stream-tin, free from columbite and containing 73 per cent. of metallic tin.

The assays were made in a naked clay pot, the charge for each being 5 grammes of cassiterite and 25 grammes of potassium cyanide. To these columbite was added, as shown below. The percentage of tin obtained in each case is calculated from the weight of the button, which, of course, includes all impurities.

SERIES I.

Ore, parts	100	100	100	100
Columbite, parts	none	10	25	50
Tin obtained, per cent	70.76	71.52	73.2	75

Condition of fire, quite hot; time, ten minutes. Character of buttons: No. 1, bright, smooth, and not very convex; No. 2, convex, dark, but not uniform in color, top rough and luster feeble; No. 3, very convex, somewhat pointed in the center, dull, no luster, but somewhat bright upon the bottom; No. 4, convex, pointed in the center, no luster even on the bottom, and color very dark gray.

SERIES II.

Ore, parts	100	100	100	100
Columbite, parts	none	10	25	50
Tin obtained, per cent	70.52	71.4	72.3	75.57

The fire in this case was but moderately bright. The buttons were the same in appearance as those obtained in the preceding series.

SERIES III.

Ore, parts.....	100	100	100	100	100
Columbite, parts.....	00	10	50	25	50
Time in the furnace.....	10 min.	10 min.	14 min.	10 min.	8 min.
Tin obtained, per cent.....	69.98	70.4	73.68	71.91	73.22

The fire was very hot. The buttons varied as in the preceding series.

Slags.—The slags had the same appearance in all cases. The addition of columbite appeared to produce but little change. The color of the slags usually inclined to yellow. The addition of 10 per cent. columbite produced in each case a more granular texture. The addition of 25 per cent. produced the strongest discoloration.

Conclusions.—Want of time prevented the making of complete analyses of the buttons and slags obtained in these experiments. It is evident, however, from the results obtained that the presence of columbite is injurious, and that the buttons owe their increase in weight to impurities derived from it. A farther examination showed that the low grade of tin obtained in the presence of columbite was due not altogether to the ordinary contamination by iron, but, in part, at least, to a definite crystalline alloy which we have separated from the buttons for farther examination. The amount of this alloy in the buttons so far examined varies from 2.5 per cent. to 10.4 per cent. of their weight. It has not been established that the alloy found in the different buttons is the same in composition; certain observations as to the form of the crystals suggest the contrary.

It was also observed that the degree of heat at which the assay was made, the quantity of columbite present, and the purity of the cyanide used had much to do with the character of the button obtained. With impure cyanide the slags were decidedly different from those obtained with c. p. cyanide, and, moreover, less of the alloy was obtained.

It is perhaps worthy of remark that with ordinary commercial

cyanide but 51 per cent. of tin was obtained from ore containing by analysis 73 per cent., while the same ore, assayed with c. p. cyanide (not, however, with the *strictly* c. p.), yielded 71 per cent. These points will form the basis of future work at the School of Mines.

NOTE ON THE NICKEL-ORE OF RUSSELL SPRINGS, LOGAN COUNTY, KANSAS.

BY FRED. P. DEWEY, WASHINGTON, D. C.

EARLY last March Mr. Jerome Coldren, an old miner and prospector, undertook a prospecting tour through the western part of Kansas, and discovered a very peculiar bed of rock, which yielded a white metal on fusion with soda. Mr. W. D. Church, city chemist at Topeka, examined the metal and found it to contain nickel. This discovery created much excitement, and was at first doubted. Many samples of the material, from a range of twenty miles, were sent to the Smithsonian Institution for examination; and in every instance a qualitative test showed the presence of nickel, although the amount appeared to vary greatly.

The ore is quite different from any other nickel-ore. It consists of a quartzose conglomerate from bean-size down, with much decomposed material between the pebbles, and the whole cemented by a more or less manganiferous limonite. In the qualitative tests, the nickel appeared to vary with the manganese. In all the tests, simple treatment with HCl dissolved a great deal of Al_2O_3.

The principal deposit is near the top of a small hill on the divide between Smoky Hill river and Butte creek. The surface-rock is the conglomerate, and, at varying distances, stains of the limonite appear and gradually increase; so that the ore is simply the lower portion of a bed of conglomerate which has been heavily charged with the manganiferous limonite. Below the conglomerate is a layer of selenite; below this, a soft sandstone, containing well-preserved fossils, and below the sandstone, a shale.

From 2 to 6 per cent. of nickel in selected samples has been reported by different chemists; but the following two analyses on large samples show the average composition of the material:

	I. Per cent.	II. Per cent.
Insoluble siliceous matter,	66.36	63.04
Copper,	0.025	0.033
Nickel,	0.258	0.260
Cobalt,	0.072	

I. Sample sent by Hon. J. J. Ingalls.
II. Sample sent by B. F. Cannon.

A company has been formed to work the ore; but, unless richer material than the above is found, the deposit is not likely to prove valuable.

A conglomerate quite similar to this is very abundant in and about the District of Columbia. I have examined several samples stained with manganese, and have always found nickel present.

THE ENGLISH VERSUS THE CONTINENTAL SYSTEM OF JIGGING—IS CLOSE SIZING ADVANTAGEOUS?

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To those familiar with ore-dressing practice, it is hardly necessary to dwell upon the importance of the jig. Within its proper sphere no substitute has been found that does the work as well or as cheaply. Its capacity is large, it requires but little attention, and the losses are small. In one form or another, it is the principal machine used in the concentration of low-grade ores, and for the separation of one metallic mineral from another; and it is almost the only machine used for the purification of coal.

The present paper is a study of the action of the jig in the light of results obtained in the Lake Superior copper mills, and at the St. Joseph Lead Co.'s dressing-works, in Missouri; and a description of experiments undertaken with the view of reproducing and isolating the conditions of jigging, in order to develop the laws which underlie the process.

There are two recognized systems of jigging, the English and the Continental or German.

The English system is a development of the hand-jigging formerly employed in Cornwall and other metalliferous districts of England, and introduced by English miners to this country. In this method the crushed ore, coarse and fine together, is first jigged on a

hand-sieve with coarse mesh; and the fine stuff, somewhat concentrated by passing through the bed of coarse mineral grains on the sieve, is again jigged on a sieve of finer mesh. In the adaptation of the system to machine-jigging, many modifications of detail have been necessary. The general treatment, however, is the same, and includes a preliminary jigging on roughing-jigs, followed by a concentration of the hutchwork on finishing-jigs of finer mesh. In both roughing- and finishing-jigs, a bed of mineral is maintained on the sieve; and the concentration is mainly effected by jigging through this bed.

The Continental or German system starts with a size-classification by screens, after which the different sizes are treated on separate jigs.

There are also combinations of the two systems. The use of a bed of mineral grains, and concentration by jigging through the sieve, has been adopted in Continental practice for fine jigging. At the Lake Superior dressing-works, a somewhat imperfect size-classification by water has been introduced as a preliminary to jigging. In the Continental dressing-works, again, the tendency of late years has been to reduce the number of sizes jigged, and to abandon the very close sizing which formerly characterized the method.

The arguments for close sizing are drawn from the well-known laws governing the fall of bodies in water, which may be stated as follows:

A body falling in still water moves at first with accelerated velocity, as in a vacuum. The resistance of the water, however, increases with the square of the velocity and finally equalizes and neutralizes the accelerating force. Thereafter the grains move with uniform falling velocity.

The laws governing the fall of regular spheres in still water may be deduced as follows:

Let d = diameter of sphere in meters.

Let δ = specific gravity of sphere.

Let V = falling velocity of sphere in meters per second.

Let p = acceleration of sphere.

Let g = acceleration due to gravity, 9.81 meters per second.

Let γ = weight of one cubic meter of water.

Let $\pi = 3.1416$.

Then, according to Weisbach's *Mechanics* (Am. Ed., p. 596),

$$p = \frac{\text{motive force}}{\text{mass}}.$$

Or, in this case,

$$p = \frac{\text{weight} - \text{resistance}}{\text{mass}}, \quad \ldots \ldots \quad (1).$$

As the sphere is submerged in water, its

$$\text{Weight} = \frac{1}{6}\pi d^3(\delta - 1)\gamma \quad \ldots \ldots \quad (2).$$

According to Rankine's *Applied Mechanics* (p. 598),

$$\text{Resistance} = \frac{1}{2}\left(\frac{\pi d^2}{4}\right)\left(\frac{V^2}{2g}\right)\gamma \quad \ldots \ldots \quad (3).$$

And, finally,

$$\text{Mass} = \frac{1}{6}\left(\frac{\pi d^3 \delta \gamma}{g}\right) \quad \ldots \ldots \quad (4).$$

From 1, 2, 3 and 4, canceling and reducing,

$$p = \left(\frac{\delta - 1}{\delta}\right)g - \frac{3V^2}{8d\delta} \quad \ldots \ldots \quad (5).$$

Whence

$$V = 1.63\sqrt{gd(\delta - 1) - d\,p}.$$

Making $p = 0$, the condition of uniform motion, and substituting for g its value, 9.81 meters, we have

$$V = 5.11\sqrt{d(\delta - 1)} \quad \ldots \ldots \quad (6),$$

the equation, as given by Rittinger and others, for the velocity of a sphere falling in still water, after it has reached the period of uniform motion.

From the discussion of this formula, it appears that the velocity of a body falling in water increases with $\sqrt{d}$ and with $\sqrt{\delta - 1}$. It also appears that spheres of different densities will fall with equal velocity, when

$$d\,(\delta - 1) = d_1\,(\delta_1 - 1)$$

or, as in the oft-cited example, the diameters of equal-falling grains of quartz and galena, during the period of uniform motion, will be as 4 to 1.

It is therefore argued that, in order to effect a separation by the water-currents of a jig, the range of size in the grains treated must not exceed the size-ratio of equal-falling grains. Larger grains of quartz, it is argued, will fall with the galena; and smaller grains of galena will be carried off with the quartz.

It has long been recognized that the conditions which obtain in jigging, are not the same as in the case of bodies falling freely in water. We have to deal not with single isolated grains, but with numbers of these grains moving together. The smaller grains move in the interspaces between the large grains, and, consequently, in constricted channels. The large grains, also, so far as their movements are independent of the surrounding grains, as in the separation of gangue from ore, and of one mineral from another, move in the interstitial channels between the other grains. That these channels must have great influence on the movement of the grains has long been recognized, and indeed was pointed out by Pernolet, one of the earliest investigators of the subject.

Nevertheless, Pernolet and those who have followed him, Sparre, Rittinger and others, have been content to assume the formula for free-falling grains as of universal application, and have drawn therefrom the arguments and data for the close and accurate sizing characterizing the Continental system of ore-dressing.

It is, however, possible to effect a very satisfactory separation on jigs, with material sized between wide limits only, and in a very imperfect manner, as in the mills of the Lake Superior copper region, or even entirely without sizing, as in the St. Joseph Lead Company's dressing-works, at Bonne Terre, Missouri.

The following experiments were undertaken with the object of determining the laws governing the movement of bodies in confined channels.

In Formula 1,

$$p = \frac{\text{weight}-\text{resistance}}{\text{mass}},$$

it is evident that the resistance only will be effected by the walls of the channel.

To determine the amount of this resistance under different conditions, a series of experiments was made with lead shot of different diameters, from 2 mm. to 16 mm., in a tube 20 mm. in diameter. The shot were suspended in the tube and subjected to rising currents of water at different velocities. The buoyant effect of the current in each case was measured by means of a sensitive spring-balance,

and its velocity, by noting the number of seconds required to fill a vessel of known capacity.

It is evident that the result will be approximately the same whether the water or the shot be moved ;* the buoyant effect in the one case, and the resistance in the other, being a function of the velocity of the current and of the diameters of the sphere and of the tube.

The use of a spiral spring-balance of wire was suggested by Prof. O. N. Rood. After experimenting with different sizes, kinds and lengths of wire, a very satisfactory balance was made from a piece of tempered steel guitar-wire, 0.01 inch in diameter and about 6 feet long, wound on a lead pencil. On releasing the spring, after winding, the coils expanded to a diameter of about 0.6 inch. This balance stretches about 0.8 inch with a weight of one gramme, and a variation of a few milligrammes in weight is readily detected. By increasing the length of wire, the balance becomes more sensitive, but its capacity is reduced by an amount equal to the weight of the additional length of wire, and the effect of slight changes in the temperature of the room is more noticeable. For the experiments with the large shot, a balance of heavier wire was used. Since the spiral spring twists and untwists somewhat as it rises and falls, the best form of index is a horizontal circular hoop of wire. A graduated scale may be used, but for these experiments it was found more convenient to measure the buoyant effects by adding weights sufficient to bring the index back to its normal position.

A constant water-supply at uniform pressure was obtained from a large tank on the floor above, and the velocity was regulated by a stop-cock as required.

Some difficulty was experienced in obtaining accurate results, especially for the smaller shot. In the first place, the velocity of the current is greatest in the center of the tube, and least at the walls, where it is retarded by friction. As a consequence, the small shot were swung to the side of the tube and showed a smaller buoyant effect than that due to the average current. It was attempted to remedy this by the use of a skeleton cage of fine wire, to which a number of shot could be fastened in any desired position. The buoyant effect of the cage, alone and with the shot, was then deter-

* The conditions are not quite the same. In the case of the suspended shot, the water impinges on the walls of the tube and on the shot with equal velocity. In the case of the falling shot, the water is at rest, so far as the walls of the tube are concerned, except at the instant of the passage of the shot. The friction on the walls of the tube will therefore be greater in the first case.

mined. This combination, however, gave results which proved to be too high. The shot, when attached to the cage, obstructed a part of the tube and, by increasing the velocity of the current elsewhere, increased the lifting-effect on the cage to an unknown extent. A string of small shot on a single hair gave better results; but the shot at the lower end of the string were swung against the side of the tube, and at high velocities the whole string was forced into this position. Weighting the string of small shot with a single heavy shot did not answer, because the variations in the buoyant effect on the large shot masked the effect on the small shot and vitiated the results. The most satisfactory data for small shot were obtained by using a cage for a sinker, so constructed as to oppose but little resistance to the water-current, and attaching the shot at long intervals to two hairs fastened to the cage-sinker at one end and the balance at the other. The buoyant effect of the cage and hairs was determined separately. Each of these attempts involved a series of comparative experiments to determine the accuracy of the method. The investigation, at first sight a simple matter, proved complex and laborious. In the course of the work, nearly six hundred experiments were made, each involving determination of velocity of current, buoyant effect and other necessary data.

The results were tabulated and carefully examined. It was found that the lifting-effect on the sphere is not proportional to the square of the velocity of the rising current in the tube, as is generally assumed, but that it is nearly proportional to the square* of the velocity in the annular space between the shot and the walls of the tube. For V^2 in the equation for the resistance to a free-falling sphere in water (Equation 3) we must therefore substitute

$$V^2\left(\frac{D^2}{D^2-d^2}\right)^2$$

in which D is the diameter of the tube and d that of the sphere.

In the case of free-falling spheres, the resistance has been found by various experimenters to be proportioned to about one-half the sectional area of the sphere.

In tubes, this apparently holds good in the case of small spheres falling in large tubes, but as the diameter of the shot approaches the full diameter of the tube, the resistance is proportional more nearly to the full sectional area than to one-half the area. The empirical expression

* More accurately, perhaps, to $V^{\frac{1}{3}}$ or $V^{\frac{1}{4}}$.

$$\frac{\left(1+\left(\frac{d}{D}\right)^{\frac{3}{2}}\right)}{2}$$

agrees very closely with the coefficients found by experiment, as is shown in the following table. The diameter of the tube was 19.5 mm. in each case:

Diam. of shot in mm.,	2.06	3.18	4.77	7.0	8.8	10.9	13.24	16.27
Diam. of tube in mm.,	19.50	19.50	19.50	19.5	19.5	19.5	19.50	19.50
Ratio, $\frac{d}{D}$.	.11	.16	.24	.36	.45	.56	.68	.83
Coefficient by experiment,	.53	.54	.57	.59	.66	.71	.78	1.00
Coefficient from formula,	.52	.53	.56	.60	.65	.71	.78	.88

For the expression

$$\left(\frac{D^2}{D^2-d^2}\right)^2\left(1+\left(\frac{d}{D}\right)^{\frac{3}{2}}\right)$$

we may substitute the simpler empirical expression

$$\left(\frac{1}{1-\left(\frac{d}{D}\right)^{\frac{3}{2}}}\right)^2$$

without introducing serious error. Substituting in Equation 3, we have

$$R=\frac{1}{2}\left(\frac{\pi d^2}{4}\right)\left(\frac{V^2}{2g}\right)\left(\frac{D^2}{D^2-d^2}\right)^2\left(1+\left(\frac{d}{D}\right)^{\frac{3}{2}}\right)\gamma \quad . \quad . \quad . \quad (7).$$

Or, approximately,

$$R=\frac{1}{2}\left(\frac{\pi d^2}{4}\right)\left(\frac{V^2}{2g}\right)\left(\frac{1}{1-\left(\frac{d}{D}\right)^{\frac{3}{2}}}\right)^2\gamma \quad . \quad . \quad . \quad . \quad (8),$$

and determining the value of V for the period of no acceleration, as before, we have

$$V=5.11\left(1-\left(\frac{d}{D}\right)^{\frac{3}{2}}\right)\sqrt{d(\delta-1)} \quad . \quad . \quad . \quad . \quad (9),$$

which is the equation for spheres moving in a tube of diameter D.

For different values of the ratio $\frac{d}{D}$ the values of V are as follows:

$$\frac{d}{D}=0.1 \qquad V=4.95\sqrt{d(\delta-1)}$$

$$\frac{d}{D}=0.25 \qquad V=4.47\sqrt{d(\delta-1)}$$

$$\frac{d}{D} = 0.5 \qquad V = 3.32\sqrt{d(\delta-1)}$$

$$\frac{d}{D} = 0.75 \qquad V = 1.79\sqrt{d(\delta-1)}$$

$$\frac{d}{D} = 0.9 \qquad V = 0.76\sqrt{d(\delta-1)}$$

If $d = 0$, or if $d = D$, then $V = 0$.

It will be seen that when the falling sphere has less than one-tenth the diameter of the tube, its velocity is but little affected. On the other hand, as the sphere approximates the diameter of the tube, its falling-velocity is much reduced, as compared with that of a free-falling body. As it is the latter condition, rather than the former, which obtains in the movements of a mass of grains on a jig-sieve, the importance of taking this retarding influence into consideration is at once evident.

The following tables give the experiments with two sizes of shot as an example of the work done. The lifting-effect computed from the formula is given in each case for comparison with that found by experiment:

Diam. of shot, .0109 m. Diam. of channel, .0195 m. Wt. of shot in water, 7.020 grms.						Diam. of shot, .01324 m. Diam. of channel, .0195 m. Wt. of shot in water, 12.580 grms.					
No. of Experiment.	Velocity of Current.	Lifting Effect Found.	Lifting Effect Computed.	Difference.	Constant from Expt.	No. of Experiment.	Velocity of Current.	Lifting Effect Found.	Lifting Effect Computed.	Difference.	Constant from Expt.
22	0.374	.976	1.003	—.027	0.69	34	0.158	.493	.474	+.019	0.81
75	0.374	.976	1.003	—.027	0.69	35	0.239	.989	1.059	—.070	0.7:
76	0.456	1.466	1.482	—.016	0.74	36	0.290	1.487	1.583	—.096	0.73
77	0.525	1.959	1.890	+.069		84	0.280	1.488	1.474	+.014	0.79
78	0.590	2.450	2.491	—.041	0.70	37	0.316	1.983	1.878	+.105	0.8:
26	0.614	2.945	2.687	+.258	0.78	38	0.357	2.478	2.419	+.059	0.8(
27	0.666	3.440	3.186	+.254	0.77	39	0.392	2.974	2.920	+.054	0.79
28	0.725	3.920	3.752	+.168	0.74	40	0.423	3.470	3.361	+.109	0.81
79	0.732	3.918	3.839	+.079	0.72	41	0.459	3.966	3.968	—.002	0.78
29	0.769	4.408	4.218	+.190	0.74	42	0.500	4.461	4.678	—.217	0.74
30	0.840	4.880	5.030	—.150	0.69	43	0.515	4.959	4.985	—.026	0.78
31	0.885	5.360	5.605	—.245	0.69	44	0.561	5.954	5.950	+.004	0.78
80	0.863	5.370	5.307	+.063	0.72	45	0.610	6.945	7.020	—.075	0.77
32	0.912	5.850	6.016	—.166	0.69	46	0.656	7.939	8.146	—.207	0.7(
81	0.912	5.850	6.016	—.166	0.69	47	0.689	8.929	8.958	—.029	0.78
33	1.013	6.815	7.321	—.506	0.66	48	0.768	10.908	11.125	—.217	0.7(
82	0.967	6.830	6.670	+.160	0.73	49	0.797	11.900	11.975	—.075	0.78
Average coefficient, 0.71 Probable error of mean, ± 0.004						Average, 0.78 Probable error of mean ± 0.004					

Sphere of Maximum Velocity.

Between the limits $d = 0$ and $d = D$ there must be a maximum value for V. From Equation 9 we find that this maximum value corresponds to the condition

$$\frac{d}{D} = 0.3968 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (10),$$

or, *The sphere having the maximum falling-velocity in any tube is the one whose diameter is four-tenths that of the tube.* Spheres larger or smaller than this fall more slowly.

Movement of Spheres en Masse.

A series of experiments was made to determine the lifting-effect of water-currents on spheres *en masse*. The experiments were made with shot of different diameters; with mixed shot; with rounded grains of galena; and with angular grains of the same mineral.

The apparatus used was a short piece of glass tubing, 20 mm. inside diameter, with a bottom of wire cloth to support the shot. This tube was attached to the apparatus previously described, so that the velocity of the rising currents of water might be controlled and measured. The velocity of the water-current was determined in each case at several different periods, as follows:

1. When the shot in the tube, through the lifting-effect of the water, began to show evidence of free movement, having a tendency to move or rearrange themselves when the tube was jarred.

2. When the shot were seen to exhibit a slight tendency to rise, being at the same time quite free to rearrange themselves when the tube was jarred.

3. When the lifting-effect was unmistakable.

When the shot are at rest they occupy 70 per cent. of the space in which they are contained. If we assume that the average diameter of the interstitial channel in which each shot moves is proportional to the cube root of the relative volumes of sphere and total space occupied, we shall have

$$\frac{d}{D} = \sqrt[3]{0.70} = 0.888. \quad . \quad . \quad . \quad . \quad . \quad (11).$$

From the observed velocities under the different conditions, the corresponding ratio of d to D was computed. We have seen

that the resistance or lifting-effect in tubes may be expressed by Equation 8:

$$R = \frac{1}{2}\left(\frac{\pi d^2}{4}\right)\left(\frac{V^2}{2g}\right)\left(\frac{1}{1-\left(\frac{d}{D}\right)^{\frac{1}{4}}}\right)^2 \gamma.$$

From this we obtain by transposition

$$\frac{d}{D} = \left(1 - dV\sqrt{\frac{\pi\gamma}{16g}\cdot\frac{1}{R}}\right)^{\frac{2}{3}} \quad . \quad . \quad . \quad . \quad . \quad (12).$$

In the experiments on shot *en masse*, R is the weight of a single shot in water, d is the average diameter of the shot, and D the diameter of the tube or channel (formed by the surrounding shot) in which the shot is supported by the observed velocity, V.

The following table gives the values computed for $\frac{d}{D}$ from the observed velocities:

Diam. of shot in millimeters.....		2.52	3.18	3.73	4.02	4.77	Average.
Shot move when tube jarred.....	Velocity.	.119	.126	.139	.146	.164	
	$\frac{d}{D}$	.902	.907	.903	.903	.904	.904
Slight lifting effect observed.....	Velocity.	.135	.131	.163	.154	.196	
	$\frac{d}{D}$	.888	.903	.886	.898	.884	.890
Noticeable lifting effect.....	Velocity.	.148	.157	.179	.171	.204	
	$\frac{d}{D}$	.877	.883	.874	.887	.879	.880
Average.....		.889	.898	.888	.896	.889	.892

The average results both for the five experiments showing a slight lifting-effect, and for the whole fifteen experiments, correspond closely with the theoretical ratio $\frac{d}{D}$ in Equation 11. Practically, the same result was obtained for the large shot as for the smaller size. The above experiments demonstrate that shot moving *en masse* follow the same law as shot moving in tubes.

Substituting the theoretical value of $\frac{d}{D}$ in Equation 9, we have

$$V = 0.833 \sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (13),$$

which is the equation for spheres moving en masse. On comparing this with Equation 6, we find that spheres moving *en masse* will fall with but one-sixth the velocity of free-falling spheres, or will require to support or lift them a rising current of but one-sixth the velocity necessary to support or lift a single isolated sphere of the same material.

Effect of Increasing Velocity of Current.

Experiments were made to test the effect of stronger currents on a mass of shot. It was found that when the velocity of the current is increased beyond that necessary to support the shot, the effect is to lift and at the same time force the spheres apart, and thus to decrease the ratio $\frac{d}{D}$ until a state of equilibrium is reached. A slight separation of the shot, with a corresponding decrease in the ratio $\frac{d}{D}$, makes necessary an increase of velocity to support the shot, in the larger interstitial channels thus formed. This explains why it is found possible to do good work in jigging, in spite of wide variations in the velocity of the rising currents employed. Two jigs, working on the same material, may be run at different speed and have different lengths of stroke and different amounts of underwater, and yet apparently do equally good work. The velocity of the rising current within certain limits simply determines how far apart the grains of mineral and of ore will be forced, and thus the height to which the surface of the bed of ore-grains will be raised. For example, by increasing the velocity five times, the volume of the ore-bed will be increased, through the forcing apart of the ore-grains, about ten times.

Rounded and Angular Ore-Grains.

Experiments were made with angular grains of galena and with rounded grains of the same mineral, from the results of which the following values were deduced:

For rounded grains of uniform size, falling en masse,

$$V = 0.490 \sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (14).$$

For angular grains of uniform size, falling en masse,

$$V = 0.536 \sqrt{d\,(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (15).$$

Large Spheres Surrounded by Small Spheres.

If we consider the case of a mixed mass of large and small spheres, it is evident that the spaces between the large spheres will be filled in part by the small spheres. If there be considerable difference in size, the small spheres will occupy 70 per cent. of this interstitial space. In such case the ratio of the volume of the large spheres to the space in which they are contained, and in which they must move, will be as 70 : 79; or 0.886. On this basis we shall have *for large spheres surrounded by small ones, when the difference in size is considerable:*

$$\frac{d}{D} = \sqrt{0.886} = 0.96 \quad . \quad . \quad . \quad . \quad . \quad . \quad (16).$$

Some experiments were made with large shot surrounded by small ones. The large shot were attached to the spring-balance and buried in the small shot, so that the spring exercised an upward tension on the shot equal to a fraction of its weight in water. The stream of water was then turned on gradually, until the large shot showed a tendency to rise under the combined action of the spring and the water-current. The velocity of the rising current was carefully and gradually increased, until the index of the spring-balance showed that the weight of the shot was just supported by the current of water. The following results were obtained:

Diam. of large shot in mm., . . .	4.77	7.00	8.80	10.50
" " small " " " . . .	2.52	2.52	2.52	2.52
Velocity of current in m. per second, .	.135	.110	.106	.104
Computed ratio, $\frac{d}{D}$, ...	.921	.926	.936	.939

It will be seen that values of $\frac{d}{D}$ were obtained varying from .92 to .94, the value increasing with the difference in diameters of the large and small shot. So far as these experiments go, they tend to confirm the theoretical ratio deduced on the supposition that the difference in diameter is large.

Substituting this value in Equation 9, we have

$$V = 0.307 \sqrt{d\,(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (17),$$

which is the equation for large spheres moving in a mass of smaller spheres, when the difference in diameter is considerable.

Mixed Spheres of Different Diameters.

Some experiments were made to test the effect of rising currents on mixed shot. Five sizes of shot, mixed in nearly equal proportions, and ranging in diameter from 2.52 mm. to 4.77 mm., were subjected to currents ranging from .091 to .154 meters per second, in a 20 mm. tube. The shot were thoroughly mixed at the beginning of each experiment.

A velocity of .091 m. was nearly sufficient to support the smallest-sized shot; these shot showing signs of motion at the top when the tube was jarred.

At a velocity of .119 m., the shot were so far supported as to be free to rearrange themselves when the tube was jarred. The small shot showed an upward tendency, especially near the walls of the tube.

With a velocity of .135 m., a slight lifting-effect was evident throughout the whole mass. The small shot showed a more decided tendency to rise toward the top. The large shot at the bottom, after separation from the small shot, were not affected by the water-current.

A velocity of .148 m. exercised a marked lifting-effect on all the shot, so long as they remained mixed. The small shot tended at once to rise toward the top, and the large shot sank. The final separation of shot of nearly the same diameter was slow. The large shot, after reaching the bottom, were not affected by the water-current.

With a velocity of .154 m., the lifting-effect throughout the whole mass was more energetic, and the separation of the large and small shot took place within a very few seconds.

The ratio of $\frac{d}{D}$ for the largest and smallest sizes of shot, computed on the supposition that the shot were supported by the observed velocity of current, are given in the following table:

Velocity of current, . .	.091	.119	.135	.148	.154
Ratio for 2.52 mm. shot, . .	.925	.902	.888	.877	.872
" " 4.77 " " . .	.947	.930	.921	.913	.910

Further experiments on a larger scale will be necessary to demonstrate how far this sizing action will be produced by the intermittent

pulsations of a jig, and what effect downward currents will produce. This sizing will not interfere with the separation of minerals of different density, but will, on the contrary, facilitate the separation of the large grains of the gangue, which will be surrounded by small grains of the heavy mineral.

This tendency of the small grains to rise does, however, interfere with the passage of the fine grains of heavy mineral through the bed of coarse mineral on the sieve. In the English method of jigging, it is a common practice to have downward currents alternate with rising currents in order to facilitate the passage of the fine mineral into the hutchwork.

Equal-Falling Spheres.

If now we determine the diameters of equal-falling spheres of quartz and galena on the basis of Equations 13 and 17, we shall have a guide to the range of size permissible in jigging these minerals. The large spheres of quartz will be surrounded by small spheres of galena, and the small spheres by spheres of equal or larger size, and we shall have for equal-falling grains, under the conditions assumed, d being the diameter for quartz, and d_1 the diameter for galena,

$$0.307 \sqrt{d\,(2.6—1)} = 0.833 \sqrt{d_1\,(7.5—1)}$$

or the diameter of equal-falling grains will be in the ratio,

$$d : d_1 : : 31 : 1 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (18).$$

We see, therefore, that if the material to be treated be sized between the limits of 1 mm. and 30 mm., it will be possible to separate the quartz from the galena. All the spheres of galena will have a greater falling-velocity than the 1 mm. grain; and all the spheres of quartz will rise more readily and fall more slowly than the 30 mm. grain. The conditions assumed are in each case those least favorable to separation.

Calculated from the formulæ for free-falling grains, the ratio of diameter for equal-falling spheres of quartz and galena would be 4 : 1.

At the St. Joseph Lead Company's dressing-works, the roughing-jigs separating galena and limestone do good work on stuff ranging from 4 mm. to $\frac{1}{8}$ mm. in size, or 32 : 1, and fairly good work on $\frac{1}{20}$ mm. stuff, with a size-ratio of 80 : 1.

The Movement of Fine Stuff in the Interstitial Channels.

There seems to be ground for an extension of the permissible size-ratio, when we consider the conditions affecting the separation of the very finest material. The size of the grains forming the mineral-bed on the jig is determined by the mesh of the jig-sieve. Grains which are small enough to move freely in the interstitial channels in this mineral-bed move under different conditions from those in the jig-bed itself.

If we consider a series of tubes, 1 mm. in diameter, as representing the interstitial channels in a bed of galena on a jig-sieve, the quickest-moving grains of quartz in such tubes will have a diameter of 0.4 mm. (Equation 10.) A grain of galena having the same velocity in a 1 mm. tube has a diameter of about 0.015 mm. (Equation 9.) It should be noted that Equation 9 gives two values of d, viz.: 0.015 and 0.915. Under the conditions, however, the largest grain of galena or quartz which can move in a 1 mm. channel will have a diameter of 0.89 mm. (Equation 11.) In these interstitial channels, then, and for this very fine stuff, under the conditions assumed, the permissible size ratio is 0.89 to 0.015 or 60 : 1.

Effect of the Irregularities of the Interstitial Channels.

A series of experiments was made with an artificial interstitial channel. A negative was made by pressing spheres, two inches in diameter, into a core of wax. The wax core was suspended in a metal tube, which was then filled with plaster of Paris. After the plaster had set, the tube was heated, melting the wax and leaving a perfect reproduction of an interstitial channel in the tube. The cross-section of this channel varies at different points. At the places where the spheres are in contact it is quite small, while between the points of tangency its area is much enlarged. With perfect spheres the maximum area of the interstitial channels is about 40 per cent. of the total area, and the minimum area about 10 per cent. The velocity of the rising current thus varies between two and a half and ten times that of the whole area of the jig-sieve. The equivalent diameters of the channel at the narrowest and widest places are about one-third and two-thirds respectively of the diameters of the spheres forming the channel. The lifting effect of a rising current in such a channel varies between wide limits. Experiments show a buoyant effect five times greater in the narrow part of the channel than in the widest place. This fact, perhaps,

explains the imperfect action of jigs on the finest stuff. The descending grains of the heavy mineral have to pass the narrow portions of the interstitial channels, where the lifting-effect is greatest; and the light grains have to be lifted through the widest places in the same channels, where the current has its minimum effect.

In practice, with angular and rounded ore-grains, the variations in the size of the interstitial channels is less pronounced than in the case of perfect spheres.

The larger ore-grains are in contact with the surrounding particles, and thus determine, to a certain extent, the size of the interstitial channels in which they move. The irregularity in the diameter of the interstitial channels will therefore affect the very finest particles only.

Effect of the Size of the Mesh of Jig-Sieves.

In the treatment of unsized material on a jig, the size of the mesh of the jig-sieve will have an important influence on the result.

1. The mesh of the sieve must bear a definite relation to the coarsest grain of ore, in order to limit the range of size treated.

2. The mesh of the sieve will determine the size of the smallest grains in the jig-bed, and, therefore, the size of the interstitial channels left by the sifting through of the finest material.

3. The mesh of the jig-sieve thus indirectly determines the extent to which fine material can be treated.

4. It is evident that grains minute enough to move freely in the small interstitial channels will be affected by the irregular size of these channels.

5. It is also evident that material of less diameter than the grain of maximum velocity, that is, grains less than four-tenths the diameter of the smallest interstitial channels, cannot, in any case, be treated to advantage.

Period of Accelerated Motion.

In the foregoing discussion, no account has been taken of the fact that the pulsating currents of a jig do not last long enough to result in uniform motion of the moving grains. The separation actually takes place during the period of accelerated motion. It has been demonstrated, however, by Sparre and others, that during this period of acceleration, the advantage is with the small grain of the heavy mineral. The short duration of the strokes of a jig is then

favorable to the separation of unsized material, and tends to increase the size-ratio permissible.

Effect of Adhesion of Water.

Since this paper was first presented to the Institute,* Mr. Luther Wagoner† has given the details of a very interesting series of experiments to determine the maximum falling-velocity of small grains in water and in glycerine. Mr. Wagoner's experiments were made in a tube 22.7 mm. inside diameter, inside of which was a second tube 6.3 mm. diameter, reaching nearly to the bottom, through which the ore-grains were charged. The current of water ascending through the 8.2 mm. annular space between the two tubes, carried upwards a portion of the ore-grains. The greater the velocity of the rising current, the coarser the grains. The velocity of the rising current was determined by noting the time required to fill a vessel of known capacity, and the grains carried up by the current were measured either directly or by computation of diameter from ascertained weight. Mr. Wagoner's experiments were on quartzite grains from 0.055 mm. to 0.815 mm. in diameter. He makes no allowance (1) for the diameter of the tube; (2) for the effect of the grains on each other; nor does he take into account (3) the fact that the observed velocity, that of the water-current, is not the true velocity of the grain which is falling against the water. The results which he has obtained will therefore, need revision.

Mr. Wagoner takes into account, however, in the construction of his formulæ, the fact that spherical bodies moving in water or air have a certain volume of water or air attached to them.‡ He assumes that such spheres have a minimum skin of constant thickness, and an outer envelope which is stripped away in some inverse ratio to the velocity.

This adhesion of the water to the spheres (and to the walls of the tube) will perhaps explain the anomalous results obtained in my experiments with the 16.27 mm. shot. In the table on page 643, it will be noted that the coefficient obtained by experiment differs widely from that computed from the formula. This difference

* Boston meeting, February, 1888. The paper was subsequently withheld for further perfecting in details; but an abstract was published in *School of Mines Quarterly*, January and April, 1888.

† Technical Society of the Pacific Coast, Meeting of March 2, 1888, *Transactions*, vol. v., p. 32.

‡ Equal to .6 the volume of the sphere. Coxe's Weisbach's *Mechanics*, p. 1031.

is even more marked in the results obtained with low velocities, when, according to Mr. Wagoner's hypothesis, the skin would be thicker.

A few of the results obtained will illustrate this. Experiments with smaller shot show the same thing, though to a less degree:

Experim'nt No.	Diameter of Shot. mm.	Diameter of Tube. mm.	Velocity of Current. m. per sec.	Lifting Effect. grammes.	Constant from Experiment.
85	16.27	19.50	.058	0.500	1.47
88	"	"	.121	2.000	1.18
91	"	"	.163	3.491	1.12
61	"	"	.238	5.990	1.05
63	"	"	.246	6.988	1.01
67	"	"	.300	9.985	0.99
71	"	"	.380	15,975	0.96
75	"	"	.477	23.964	0 92

The constant from the formula is 0.88.

As the effect of this film of water tends to disappear with higher velocities, sufficient to support or raise the sphere or grain of mineral, it has not been thought best to complicate the formulæ already given by introducing terms to provide for the increased diameter of the grains due to this film of varying thickness, especially as the formula here deduced seems to apply so well to the movement of grains *en masse*. In Mr. Wagoner's experiments with small spheres and low velocities, this film surrounding the grains seems to play a much more important rôle.

Practical Illustration.

The roughing-jigs in the mill of the St. Joseph Lead Co.,* at Bonne Terre, Mo., treat unsized stuff from 4 mm. to the finest slime.

The pistons of these jigs make 150 strokes, 2″ long, per minute. The area of the piston is about one-third the area of the jig-sieve. The amount of under-water is but little in excess of that discharged with the hutch-work, and has but little effect on the velocity of the jig-currents. The rising current is therefore that due to the movement of the piston, or about 0.08 m. per second. This, according to Formulæ 14 and 15, will raise a 4 mm. grain of galena, the maximum sized grain of the galena-bed. This furnishes a crucial test of the

* See also a paper on "The New Dressing Works of the St. Joseph Lead Company," by the writer, at page 659 of the present volume.

accuracy of the formulæ and indicates that they can be used in practice to compute the stroke of jigs if desired. The downward current has about the same velocity. The jig-sieves are of No. 6 wire-cloth, with about 2.8 mm. openings. The grain of maximum velocity is one-eighth millimeter. This was determined by observing the relative amounts of each size passing through the sieve, and going over the sieve, respectively. Tailings finer than this show free galena, though the loss is not serious above one-twelfth or, perhaps, one-sixteenth millimeter. The hutch-work begins to be poor, indicating imperfect concentration, at about one-quarter millimeter.

The finishing-jigs at the same mill treat stuff one millimeter and less in size. The jigs are run at 270 strokes, one-quarter inch long, per minute. The jig-sieve is of No 8 wire-cloth (2 mm.). These jigs treat successfully stuff one-twentieth mm. in size.

It will be seen that these results tend to confirm the laws developed by this investigation. The concentration is quite satisfactory, and there is no loss in free galena except in the very finest material, stuff fine enough to be affected by the irregularities in the smallest interstitial channels;* and so fine as to be smaller than the grain of maximum velocity.

The range of size treated successfully on these jigs, without loss of free galena in the tailings, is from 4 mm. to $\frac{1}{8}$ mm. in one case, and 1 mm. to $\frac{1}{20}$ mm. in the other; or 1 : 32 and 1 : 20 respectively.

The English Method of Jigging.

The principal advantage of the English method of jigging is that it dispenses with the operation of sizing by screens. The plant is thus simpler and cheaper, and the expense of treatment less. If the jigging be conducted entirely without size-classification, as at Bonne Terre, it has the further advantage that very fine material can be treated successfully on jigs, which would otherwise have to be treated on tables. The bulk of the gangue, both fine and coarse, is thus at once separated by the "roughing-jigs," leaving but a small amount of rich stuff to be treated on the finishing-jigs and tables. As the treatment of the fine stuff by itself is troublesome, and the capacity of slime-jigs and tables is small, the plant and method of treatment are still further simplified and cheapened.

* In computing the probable size of these smallest channels, it must be remembered that the stuff sifts slowly through the jig-sieve, so that there is always material in the bed much finer than the mesh of the wire-cloth.

The English method is especially well suited to the concentration of low-grade ores on a large scale.

Enrichment of Hutchwork.

The principal objection in practice to the English system of jigging is the imperfect concentration of the material passing through the jig-bed. The hutchwork is much richer when the stuff is sized before jigging. This difficulty is overcome at Lake Superior and elsewhere by the re-treatment of this hutchwork on finishing-jigs. At Bonne Terre, the hutchwork is classified in pointed boxes, the sands re-treated on jigs, and the slimes concentrated on side-bump tables.

The present investigation suggests means whereby the hutchwork can be enriched without the necessity for this second treatment. By treatment of fine and coarse material together on the same jig, interstitial channels will be formed between the coarse grains, in which the fine stuff can be very perfectly concentrated. Again, by decreasing the mesh of the jig-sieve the size of the interstitial channels may be reduced, and still finer material successfully concentrated. Experiments made two years ago, at one of the Lake Superior mills, to test the possibility of reducing the number of grades of sand, developed the unexpected result that the hutchwork was much improved by jigging coarse and fine stuff together, the reason for which is now clear.

At Bonne Terre, also, experiments have proved the same thing. At this mill, however, the problem is complicated by the necessity of running the roughing-jigs so as to suck as much fine stuff through the sieve as possible.

Treatment of Slimes.

The problem to be solved in connection with the English method of jigging is to determine the best method of treating the stuff too fine to be concentrated successfully in the interstitial channels of the jig-bed.

The first and most obvious method is the one adopted at Lake Superior; viz., to separate the fine stuff by water-classification before jigging the sands.

The objections to this course are, first, the quantity of water required to effect the separation of the slime; and second, the fact that much fine stuff will be sent to the tables that could be treated successfully on the jigs.

A second plan is to separate the fine stuff from the tailings of the jigs by proper classifiers. This is perfectly feasible, and has the advantage that the jigs can be so run as to produce a rich hutchwork. This plan requires large settling-tanks, and a considerable amount of clean water to effect the separation of the slime. It has the advantage over the plan of separating before jigging that the fine mineral is partly concentrated and saved by the jigs.

A third plan, which will effect a partial solution of the problem, is to run the jigs so as to take full advantage of the interstitial action. By reducing the mesh of the jig-sieve finer material can be concentrated. With a given maximum size for the coarse sand-grains, however, a limit will be found, beyond which the mesh of the sieve cannot be reduced. Possibly it may be found practicable to use a three or four-sieved jig, with different mesh on each sieve, and, by varying the stroke of the pistons, to adapt each sieve to the saving of a certain grade of sand. At Lake Superior the tail-sieves have finer mesh than the head-sieves. Or the roughing-jigs may be run with little under-water, so as to suck all, or nearly all, of the fine stuff through, thus insuring poor tailings, and the hutchwork may then be treated again on jigs of finer mesh. These finer jigs should be run in the same way, and their hutchwork should be treated on tables.

By one or another of the above methods, it may be found practicable to save and treat the slime without using classifiers to separate it from the sands. There is certainly a large field here for careful and systematic experiment.

A fourth plan is to conduct the crushing so as to produce a minimum amount of slime: for example, by a system of gradual crushing, using two or more sets of rolls; or by coarse-crushing, followed by jigging and fine-crushing of "raggings" only; or, again, by calcining the ore so as to make it more friable. By one or another of these methods it may be possible to limit the production of fine stuff, so that in many cases it may be allowed to escape without serious loss.

Résumé.

The following are the main points developed by this investigation.

1. Bodies falling through water in a tube do not attain as high a velocity as in falling through the same medium in large vessels.

2. The falling velocity is but little affected when the diameter of the body is less than one-tenth that of the tube.

3. The falling velocity is the more retarded as the diameter of the body approximates that of the tube.

4. A sphere four-tenths the size of the tube will develop the greatest falling velocity, and will require a current of maximum velocity to support or raise it.

5. Grains falling *en masse* are really moving in confined channels, and follow the law of the movement of bodies in tubes. The falling-velocity, and the velocity of the current necessary to support or raise the mass of grains, increase and diminish with the distance apart of the grains.

6. The diameter of the channel in which the single grain moves equals the cube root of the volume of the grain with its proportion of the interstitial space, or, in other words, the cube root of the space occupied by the grain.

7. In a mass of grains of different sizes, the large grains move relatively in smaller channels than the small grains. The ratio of the diameters of equal-falling grains of quartz and galena, under such conditions, is 31 to 1, instead of 4 to 1, which latter ratio holds good for free-falling grains only.

8. The formulæ for grains moving in tubes, when applied as above to grains moving *en masse*, enable us to compute the velocity of jig-currents and thus determine the proper length and number of strokes of the jig-piston. The old formulæ gave results many times too large.

9. *The present investigation demonstrates that close sizing is not necessary for the separation of different minerals by jigging, unless the difference in specific gravity is small.* This has been recognized in practice for some time, especially in this country.

10. Downward currents are apparently necessary to success in jigging through a bed. This requires confirmation by experiments on a larger scale.

11. Very fine material, less than $\frac{1}{10}$ millimeter in diameter, can be treated successfully on jigs, *if treated with coarse stuff*, the concentration taking place in the small interstitial channels between the grains forming the mineral bed. *For the treatment of fine stuff on jigs, close sizing is a positive disadvantage.* Jigs work well on mixed stuff, and very badly on fine stuff alone. Stuff less than four-tenths the size of the smallest interstitial channels cannot be treated successfully in this way.

12. The size of the mesh of the jig-sieve has a very important influence, and must be proportioned to the work to be done.

13. The English method of jigging without sizing, except possibly so far as is necessary to remove the very finest slimes, has many advantages, and should be more generally adopted.

This investigation has proved, to the writer, exceedingly fruitful of suggestions of experiments to be made and of further truths to be developed. The results so far attained are here presented, in the hope that they may prove equally suggestive to others interested in this important branch of mechanical metallurgy. The generally accepted ideas on the subject of ore-dressing apparently need careful revision.

THE NEW DRESSING-WORKS OF THE ST. JOSEPH LEAD COMPANY, AT BONNE TERRE, MISSOURI.

BY H. S. MUNROE, SCHOOL OF MINES, NEW YORK CITY.

(Buffalo Meeting, October, 1888.)

THE dressing-works of the St. Joseph Lead Company were destroyed by fire, February 26th, 1883. Within about four months, or on July 5th, 1883, the new mill, with a capacity of 500 tons per day, was built and in active operation. The capacity of the new mill has since been increased to 800 tons per day.

The old mill was built in 1867, by Mr. C. B. Parsons, the present superintendent. It was a curious rambling old structure, which had grown with the demands of the business; a few jigs having been added in one place, a table or two in another and wings and sheds added to the main buildings, from time to time, as it was found necessary to increase the efficiency or the capacity of the mill. The old mill was, moreover, a most interesting place to the student, a perfect "museum of ore-dressing." Nearly every form of dressing apparatus known to the art had, at one time or another, found a place under its roof, jigs of many kinds, concave, convex, side-bump and stationary tables, tyes, buddles, dolly-tubs, etc., etc. The method of dressing followed in the main the lines of English practice, but with many modifications, both in the machinery and the methods of working, due to the inventive genius of Mr. Parsons. Different forms of crushing apparatus were tried, and the best method of handling the ore and the different products of

the mill, was carefully worked out. At one time, moreover, the mill was surrendered to a German expert, of acknowledged ability, and the Continental system of dressing was given a fair trial, with its numerous sizing-drums and jigs with varying speed and stroke adapted to each ore-class.

The new mill represents, then, the results of fifteen years' study and experiment in the old structure, and is a shining example of the "survival of the fittest" both in apparatus and in methods of treatment.

The salient features of the new mill are: first, the method of treatment, which follows English rather than German precedents; second, the dressing apparatus, which has many novel features; and third, the construction and arrangement, and the methods of handling the ore and mill-products, which are quite unique.

The work of dressing is carried on at Bonne Terre under special difficulties. For many years the only water-supply has been the rainfall of a basin about a square mile in area, the water being collected and stored in a reservoir formed by a dam across the little valley. The water is used over and over again, the portion heavily charged with slime being allowed to escape into a small settling pond, from which it is again pumped into the mill. The only water lost is that carried off in the wet sand and that lost by evaporation. This loss is made good by drawing small quantities from time to time from the reservoir.

The disposal of the waste sands or "chats" is also a matter of some difficulty. Until recently the company had no dumping ground available near the mill, which fact, with the lack of water, made it impossible to run off the sand by launders in the usual way. Resort therefore was had to the plan of carrying off the sands in railroad cars designed for the purpose. In the old mill these cars were loaded by bucket-elevators, but in the new mill special "chat-tanks" have been introduced for the purpose. The water which drains from the cars runs into the settling-pond and is saved. The "chats" are used for grading, for railroad embankments and ballast, and recently in the construction of a new dam made necessary by the undermining of the old reservoir. The bulk of the sands are dumped in the valley above the dam, so that the water which they still retain drains out and eventually finds its way into the reservoir.

By a recent consolidation, these disadvantages have been removed in part. The company now owns a large pumping-plant on Big river, about three miles away, and only the additional water-supply

so gained has made it possible to continue work through the severe drought of the past season. A more convenient dumping-ground for the waste sands, also has been secured.

The ore is galena, associated with some iron pyrites carrying traces of nickel and cobalt. The mineral is disseminated in a magnesian limestone in irregular segregated deposits of large size. Formerly it was supposed that these deposits occurred in definite "ranges" and at certain horizons only, but further exploration with the diamond drill has proved the existence of ore-bodies over the whole area prospected, nearly a square mile, and through a thickness of over three hundred feet of limestone. The deposit is worked by enormous chamber-workings, supported at intervals by heavy pillars. Underhand stoping is employed and the ground is broken with machine-drills and high explosives. Formerly small diamond drills were used for boring the blast-holes, and very deep holes, thirty feet and more, and heavy charges of explosive were employed. The recent advance in the price of diamonds has made necessary the introduction of percussion-drills; and the excessive amount of block-holing, incident to the large blasts, has caused the employment of blast-holes of more moderate depth. Owing to the large size of the workings, the cost of breaking ground is quite low. The rock is loaded in small iron cars, hauled by mules, and brought to the surface through vertical shafts.

The main shaft is close to the mill, and the cars are run by the launders into the dumping-cradles above the crushers. The mine-cars from the more distant shafts are brought to the mill on special platform cars by switching-engines. These platform cars hold twenty mine-cars each, and there is a sufficient number of them to avoid delay.

Outline of Method of Treatment.

The method of treatment may be outlined as follows: The ore is crushed by jaw-crushers and rolls, and screened dry through a 6 mm. screen. The sands passing through the screen are thoroughly mixed with water and elevated by centrifugal pumps to distributors and divided among the Parsons jigs, without any previous sizing or classification. The tails ("chats") after passing over the two sieves of these jigs receive no further treatment, and are conveyed by launders to the "chat-tanks." Coarse galena and raggings are skimmed by hand from the jigs at intervals, leaving always a sufficient bed to ensure good hutchwork. The hutchwork which comes

through the sieves of the Parsons jigs pass through a series of spitzkasten. The heavy galena, mixed with some sand and slime, settles in the first box of the series from which it is fed to a trunking-machine. The pure galena from this machine falls into railroad cars and goes to the smelting-works. The tails from the trunking-machine, together with the sands settling in the second box of the spitzkasten, are elevated by centrifugal pumps and divided between the Hartz three-sieved jigs. The tails of the Hartz jigs receive no further treatment, going directly to the chat-tanks. Galena and pyrites are skimmed from the sieves of these jigs. A bed of galena is, however, maintained on all three sieves so as to ensure a rich hutchwork. The hutchwork of these finishing-jigs is nearly pure galena, and goes to galena-boxes on the lower floor, which are emptied from time to time, and the galena loaded on cars to go to the smelting-works.

The fine slimes settling in the third and fourth boxes of the spitzkasten are united and raised by centrifugal pumps to the distributors feeding the first row of Parsons-Rittinger tables. The middlings from these tables are treated on the second row of tables. The tails from all the tables flow into the chat-tanks, and the heads run into galena-boxes on the lower floor from which they are loaded into cars.

The raggings, containing from 12 to 20 per cent. of lead, which are skimmed from the Parsons jigs, are recrushed by fine rolls and elevated without screening to a line of Hartz three-sieved jigs. These raggings contain considerable pyrites.

Arrangement of the Mill.

Plate I. gives a general plan of the mill and Plate II. a transverse section. The mill is a two-story structure. On the main floor are the ore-bins, the roughing-jigs, the finishing-jigs, and the tables. All the work requiring abundance of light and careful supervision is concentrated on this floor. The entire absence of overhead machinery, screens, shafting and belting, classifying-tanks and launders is very noticeable. There is nothing to intercept the light falling on the jigs and tables, and the whole floor is so open that supervision is rendered easy and effective. The roof is a mere umbrella of corrugated iron, with light iron trusses and supported on slender columns. All the heavy weights and jarring machinery are as near the ground as possible. The rolls, screens, centrifugal pumps, spitzkasten, chat-tanks and galena-boxes are on the lower floor near the

Plate 11.

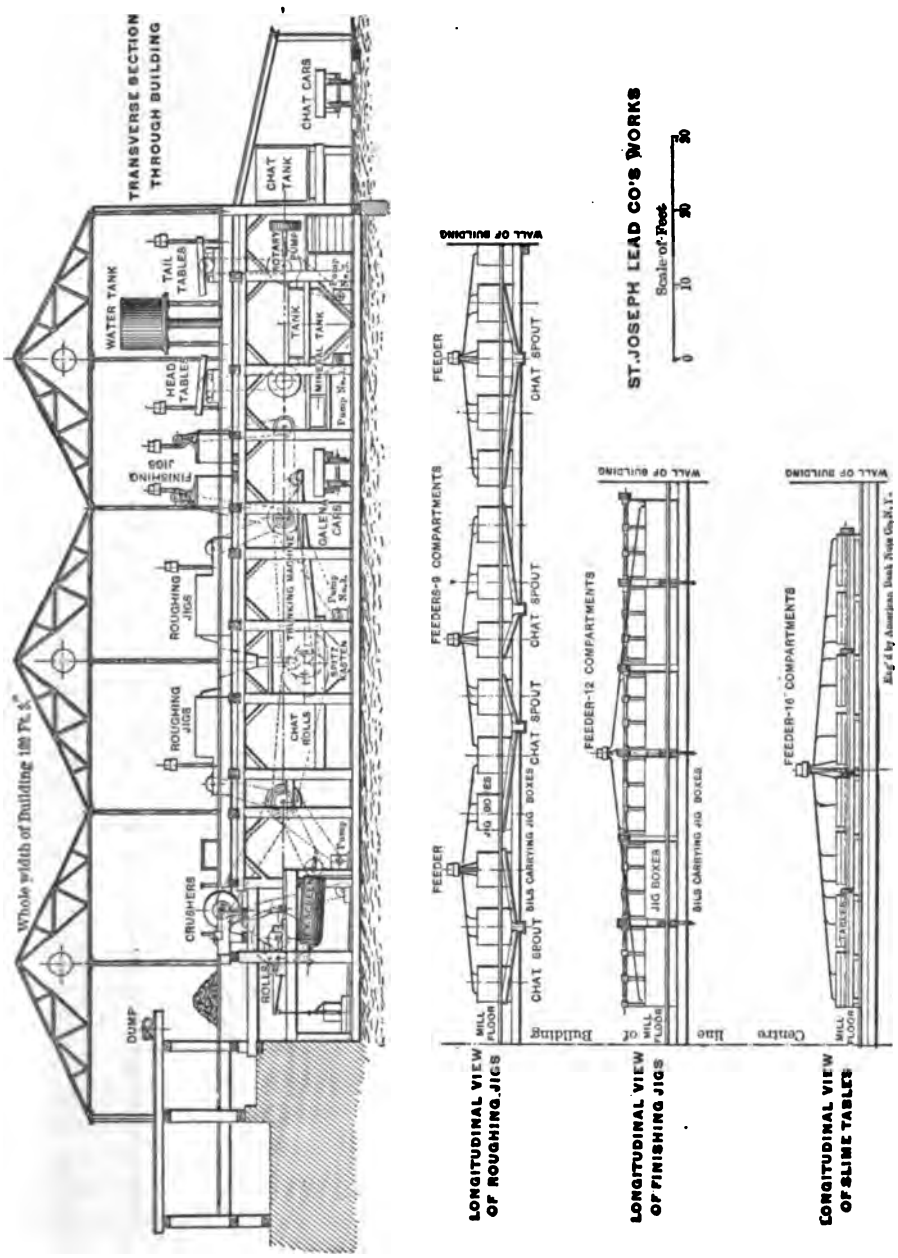
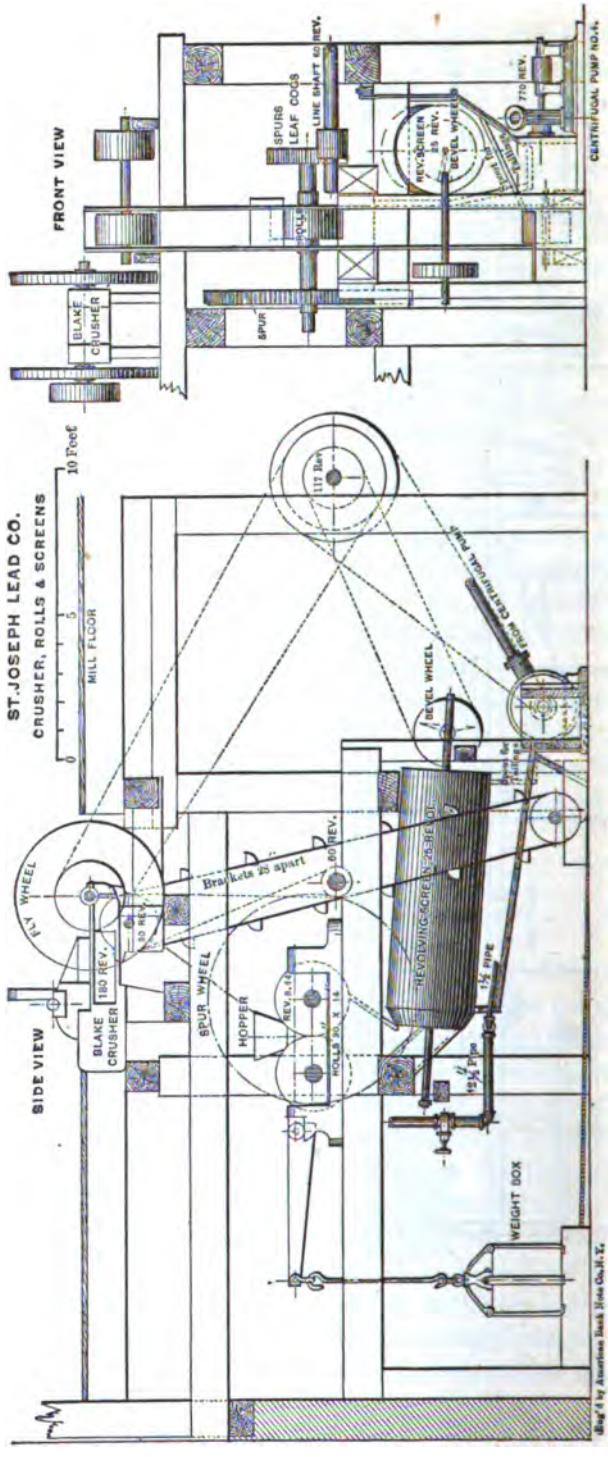

TRANSVERSE SECTION THROUGH BUILDING
Whole width of Building 180 Ft. 6"
WATER TANK
TAIL TABLES
HEAD TABLES
FINISHING JIGS
ROUGHING JIGS
ROUGHING JIGS
CRUSHERS
DUMP
ROLLS
CHAT TANK
CHAT CARS
ROTARY PUMP
MINERAL TANK
GALENA CARS
TRIMMING MACHINE
CHAT ROLLS
BOTT. JIGS
LONGITUDINAL VIEW OF ROUGHING JIGS
FEEDER
FEEDER-9 COMPARTMENTS
WALL OF BUILDING
CHAT SPOUT
BUB CARRYING JIG BOXES
MILL FLOOR
Building
LONGITUDINAL VIEW OF FINISHING JIGS
FEEDER-12 COMPARTMENTS
JIG BOXES
Line of
LONGITUDINAL VIEW OF SLIME TABLES
FEEDER-16 COMPARTMENTS
Centre
ST. JOSEPH LEAD CO'S WORKS
Scale of Feet
0
10
20
30

Plate III.


ST. JOSEPH LEAD CO.
CRUSHER, ROLLS & SCREENS
FRONT VIEW
SIDE VIEW
MILL FLOOR
10 Feet
BLAKE CRUSHER
FLY WHEEL
180 REV.
130 REV.
SPUR WHEEL
HOPPER
Brackets 25 apart
60 REV.
REVOLVING SCREEN 25 REV.
BEVEL WHEEL
CENTRIFUGAL PUMP
1½ PIPE
WEIGHT BOX
SPURS
LEAF COGS
LINE SHAFT 60 REV.
PULLEY
SPUR
770 REV.
CENTRIFUGAL PUMP NO. 4.

ground. The shafting, gearing, pulleys and belting are all below the main floor.

This arrangement has many advantages. It is cheap; the heavy framing being confined to one story. It is strong and durable; the jar of the moving machinery, crushers, jigs, and tables having but little effect on such a low, flat and strongly-braced structure. All the classifying- and settling-tanks, with the enormous weight of water which they contain, rest directly on the ground. Finally, the absence of overhead machinery and apparatus, besides increasing the light and facilitating superintendence, lessens materially the risk from fire, there being absolutely nothing to burn above the main floor of the mill.

The main disadvantage attending the arrangement of the mill in two floors, as described, is the necessity for elevating the material to be treated several times. This is accomplished, at Bonne Terre, by the use of centrifugal pumps by which the sands are elevated *with the feed-water* required for the proper working of the various jigs and tables. It should be noted, moreover, that the bulk of the material is elevated but once. After passing over the roughing-jigs, 600 tons of waste sand go at once to the chat-tanks. Of the remaining 200 tons, 74 tons are mineral and raggings, and 20 tons escape with the overflow of the spitzkasten, leaving only 106 tons to be elevated again. The average lift of the 800 tons raised by the centrifugal pumps, including that which has to be elevated a second time, is less than thirty feet. If a mill of several stories like an anthracite breaker, or one arranged in steps or terraces like the Lake Superior mills, had been built at Bonne Terre, it would have been necessary to deliver the ore at a level at least forty feet higher than at present, involving a much more expensive building, and increased cost of hoisting, forty feet instead of thirty, for the whole 800 tons, merely to save the elevating of 106 tons a second time.

Again, if the mill had been built in several stories both the wash-water and feed-water for the different machines would have to be raised about twenty feet higher than at present, and as twenty-nine tons of water are required to treat one ton of ore, this additional lift would be a serious matter.

In round numbers the saving by the present arrangement amounts to about 520,000 foot-pounds per day, or over 20 horse-power.

Lighting of the Mill.

The mill is lighted by numerous large windows in the side walls, and by nearly one hundred large skylights of heavy glass inserted in the roof. At night the necessary light is furnished by numerous incandescent electric-lamps, those near the feeding platforms, which are liable to be struck by flying bits of ore, being protected by heavy wire-cloth.

Crushing of Ore.

Plate III. shows the general arrangement of one set of crushing and screening apparatus. There are ten such in the mill, each crushing about 80 tons in twenty-four hours, consisting of a 7″ x 15″ Blake crusher, lever pattern, with corrugated jaws; a pair of Cornish rolls, 14″ x 30″, with chilled tires; a revolving screen, 3 feet diameter by 8 feet long, with 6 mm. perforated steel-plates, a bucket-elevator for the material coarser than 6 mm., and a centrifugal pump.

The jaw crushers are set to crush to about 38 mm., or 1½ inches, and the rolls to crush to about 16 mm., or ⅝ inch. Of the material passing through the rolls at any time, about one-third is too coarse to pass through the screen, consequently the ore passes through the rolls one and a half times. While the perforations of the screens are 6 mm. in diameter, not more than 2 per cent. of the screened product is coarser than 4 mm.

The crushers are fed by hand, one man to each machine. Care is taken to have the feeding regular, as on this depends the amount of material going over the jigs, tables, etc., which must be kept uniform.

Some interesting data were obtained as to the fineness of the ore at different stages of crushing, samples having been sifted through sieves of different mesh and the percentage of each size computed.

Size.	Average of Ore from Mine.	Product of Blake Crusher, crushing.		Product of Rolls.	Screened Ore. (Average.)
		Rich Lumps.	Lean Nut Ore.		
Above 38 mm.	61.3	.....	.....	.....	.....
16 mm. to 38 mm.	23.2	67.7	75.2	.....	.....
4 mm. to 16 mm.	7.5	20.7	18.9	35.2	2.0
1 mm. to 4 mm.	4.9	8.6	4.4	} 64.8	41.0
¼ mm. to 1 mm.	1.6	.8	.4		27.0
Below ¼ mm.	1.5	2.2	1.1		30.0
	100.0	100.0	100.0	100.0	100.0

It will be seen that of the ore coming from the mine, nearly 40 per cent. is as fine as if it had passed through the jaw-crushers, 15.5 per cent. is as fine as though it had passed through the rolls, and 8 per cent. is fine enough for the jigs. This latter portion is very rich, containing over 20 per cent. of lead, while the average richness of the ore is about 8 per cent.

Two experiments were made with the Blake crusher; the first with large lumps of rich ore, the crushing of which should give a maximum of fine stuff; and the second with small pieces of lean ore, which should yield a minimum of fine stuff. The jaws were set to crush to 38 mm. or 1½ inches. About one-third the product in first case, and one-quarter in the other was as fine as though it had passed through the rolls. But little fine stuff, less than 1 mm., was made in either case.

The figures in the last column show the average composition of the crushed ore after it has passed through the 6 mm. screen, and as it goes to the jigs. It will be seen that the rolls produce a large proportion of fine stuff. The losses in the treatment of this fine stuff are very great. Its production could, perhaps, be lessened somewhat by some changes in the arrangement of the crushing-machinery, but it is evident that some better form of crushing-apparatus than rolls, is urgently needed for such fine comminution as is required at these works.

Of the 30 per cent. of stuff, less than ½ mm., there is produced:

In the mine,	1.6 per cent.
By the crusher,	.8 per cent.
By the rolls,	27.6 per cent.
Total (as above),	30.0 per cent.

The experiment of using coarser screens than 6 mm. has been tried without success, the losses from included mineral being much larger. It is proposed to make a trial of other forms of crushing-apparatus, the results of which will be of great interest. Since a machine for fine crushing, that will produce a minimum of slime, of large capacity and economical, is much needed, a promising field is open to inventors.

In crushing 224,203 tons of rock in 1886–1887, an average of 8 sets of crushing apparatus being in operation, the following new parts were required:

For the crushers:	
12 levers at $25.00,	$300.00
9 jaw-plates at $15.50,	139.50
12 jaw-plates at $12.00,	144.00
Toggles, check-plates and sundries,	247.80
Total,	$831.30

or an average of about $100 for each crusher. This does not include the cost of babbitting the bearings nor labor in making repairs.

For the rolls:	
7 pairs tires at $120.00,	$840.00
Gear-wheels and pinions,	335.00
Total,	$1175.00

or about $147 for each pair of rolls. The tires of the rolls used for coarse crushing are not turned when worn, but are replaced by new ones. The tires of the fine rolls are kept in good condition.

For the screens, 21 sets of perforated plates @ $60.75, equal to $1275.75, were required, or an average of 2.6 sets per year for each screen.

The average life of the wearing parts of a jaw-crusher is therefore about 8 months; each set of chilled tires on the rolls lasts about a year; and a set of screen plates about 4 months.

Centrifugal Pumps.

The centrifugal pumps, used for elevating water and sand together, are made under the Heald & Cisco patents, by the Morris Machine Works, Baldwinsville, New York. They are made somewhat heavier than for ordinary work, the metal in the fan, for example, being ¾-inch thick. No attempt is made to repair the pumps. Usually the fan and shell are worn out in about the same time. The work of replacing an old pump by a new one takes an hour to an hour and a half. When the centrifugal pumps were first used for pumping coarse sands, a new pump lasted about three weeks, but by increasing the thickness of the wearing parts the life has been prolonged. It is possible that by lining the pumps with rubber, as is sometimes done in the case of centrifugal machines used for dredging, the pumps might be made still more durable. The average life of one these pumps is 100 days.

The smaller pumps (3-inch) used for elevating the fine sands and slimes last much longer.

An incidental advantage from the use of centrifugal pumps is the very thorough mixing of water and sand, a matter of some importance, because the ore is crushed and screened dry and only mixed with water just before entering the pump. By this thorough mixing the percentage of float-mineral is undoubtedly diminished.

The sand and water is conveyed to the distributors feeding the jigs by 4-inch extra-heavy gas-pipe, which lasts about two years. Any change in direction is made by bending the pipe with as long and gentle a curvature as possible. The 3-inch pipes, conveying the fine sands and slimes, have been in use over four years, or since the building of the new mill, and are apparently still in good condition.

The Parsons Feeder.

A simple and effective device, shown on Plates IV. and V., is used to subdivide the sands or slimes, elevated by a single pump, among a number of machines, jigs or tables. A satisfactory method of doing this, so as to secure an equal and uniform distribution, without any concentration or classification of the sands on one machine at the expense of the others, has been for a long time a great desideratum in dressing works. The ingenuity of Mr. Parsons has furnished a very satisfactory solution of this difficult problem. The feeder or distributor is a casting at the top of the pipe from the centrifugal pump, divided by partitions into a number of radial boxes, which surround the delivery pipe and are connected therewith, by vertical slots of uniform size. By increasing the number of radial partitions the streams of sand or slime can be subdivided to any desired extent. From the bottom of each of these boxes a pipe conveys the sand or slime to the jig or table.

When it is necessary to shut off the supply temporarily from one jig or table, its box in the distributor is filled and the opening closed with a piece of cloth, which for convenience in handling, is fastened to an iron rod with a loop-shaped handle. The distributors are made of cast iron, and are quite durable. Those now in use have required no repairs since the mill was built, 4½ years ago. The pipes leading from the distributors to the jigs must be renewed occasionally.

Roughing-Jigs.

Plate IV. shows the details of the Parsons jig, used for the preliminary jigging. It is a two-sieve under-piston jig, each sieve 24″ x 39″ (or 22″ x 37″ in the clear.) The piston is vertical, 15½″

in diameter, and moves in a short horizontal cylinder, 4½″ long, in the partition between the two jig-boxes. The piston-rod is horizontal and enters through a stuffing-box. For convenience the jigs are made double, *i.e.*, four sieves or two jigs, are united in one machine. One great advantage of this form of jig is the small floor space occupied by a large number of machines, the whole area of the jig-box being available for jigging. The stroke of the piston is 2 inches, and the number of strokes per minute one hundred and fifty. The quantity of under-water required is about two cubic feet or 16 gallons per minute. The consumption of feed-water is 16 to 18 gallons per minute, containing about 13 pounds of sand (dry weight). There are nine jigs, 4½ machines, for each set of crushing apparatus. Each jig treats, therefore, about 9 tons per 24 hours, or less than one-third the capacity of a jig of equal area according to data given by Rittinger, so that the mill has ample jig capacity.

In the Lake Superior copper mills the tendency, of late years, has been to increase largely the number of jigs, in the attempt to lessen the losses in the tailings. The St. Joe mill has, however, a larger jig capacity, in proportion to the rock crushed, than any of the Lake Superior mills. For the treatment of 800 tons of material per 24 hours there are 252 jig-sieves. The best equipped mill in the Lake Superior region would have on the same basis, but 224 sieves. The rough sands at the St. Joe mill are divided between 90 sieves, or 9 tons per 24 hours to each. In the Lake Superior mill alluded to there would be but 60 sieves for the same mill capacity. At Lake Superior, however, about 45 per cent. of the ore is separated in the classifiers as slime, and goes directly to the tables. This reduces the average amount of sand treated on each jig to 7½ tons per 24 hours. On the other hand, the sands are very unequally divided, so that some of the jigs treat over 10 tons per 24 hours, and others as little as 5 tons. These latter jigs treat very fine stuff, and at 5 tons per 24 hours are really worked to their full capacity. The loss on these jigs is fully as large as on the coarse jigs, notwithstanding the greater fineness of the material, and the smaller quantity of included copper present.

Experiments made by the writer, about two years ago, at one of the Lake Superior mills, seemed to indicate that the capacity of the mill could be increased fully 50 per cent., and this without increasing the losses, simply by making but two grades of sand in the classifiers instead of four. The practice at the St. Joe mill, where no attempt is made to classify the material before jigging, and where,

consequently, sands and slimes are treated together on the same jigs, represents a further and extreme step in the same direction. The results obtained by this novel procedure are opposed to our preconceived ideas and accepted theories on the subject of jigging, and the St. Joe practice is certain to have important influence in modifying jigging methods in the future. The results obtained, it is true, are not wholly satisfactory, the loss of galena in the very finest slimes being large. In other respects, however, the method has many advantages, and the loss in the fine stuff can be brought within reasonable limits by simple modifications in the methods of treatment, which will be discussed further on.

During the past year the writer has had the opportunity of studying very carefully the working of these jigs. A large number of samples were taken, and several hundred assays made under his direction. In order to determine the action of the jigs on the coarse and fine grades of sand and slime, samples of the material treated, of the tailings, and of the hutchwork were subdivided by sifting, and the different portions assayed separately. From the results of the sifting also and from other data, it was possible to estimate the proportion of sand and slime of different grades passing through the jig-sieves into the hutchwork, and the proportion of each size which passed over the sieve and into the tailings under different conditions.

The following table contains typical assays of the material treated and of the products of jigging:

Size.	Percentage of sand of each size.	Lead in Ore. Per cent.	Lead in Hutchwork. Per cent.	Lead in Raggings. Per cent.	Lead in Tails. Per cent.
1 mm. to 6 mm.....	41.1	6.32	.....	16 20	1.06
½ mm. to 1 mm.....	29.6	9.10	74.0	7.97	.96
⅛ mm. to ½ mm.....	9.3	13.81	19.2	....	.71
1/24 mm. to ⅛ mm.....	1.5	12.93	14.8	....	1.09
1/60 mm. to 1/24 mm.....	2.2	7.84	8.8	....	1.74
Below 1/60 mm.....	16.3	12.22	16.4	.....	6.07
Average.....	100.0	8.93	22.3	16.54	1.53

From the assays of the different portions of the ore it will be seen that the stuff less than ½ mm. is richer than the average, while the portions coarser than ½ mm. are poorer. This is due to the friability of the galena, which is crushed finer than the gangue.

The hutchwork is quite poor, containing but 22.3 per cent. of

lead. This is due to the large proportion of partially concentrated fine stuff, less than $\frac{1}{8}$ mm., about two-thirds of which passes into the hutchwork. This fine stuff is separated by the trunking-machine and the pointed-boxes, and concentrated on the finishing-jigs and tables.

The assays of the several portions of the jig-tailings show very closely the limits within which the jigs are doing good work. It will be seen that the amount of lead in the tailings decreases with the fineness of the sands down to $\frac{1}{8}$ mm. A careful examination of these coarser sands will fail to show any free mineral, the lead being present as included galena. The finer the comminution the smaller the amount of such included mineral, and consequently the poorer the sands. Below $\frac{1}{8}$ mm. the tailings increase in richness, which is due to the presence of free galena. The loss from this cause is not serious, save in the portion finer than $\frac{1}{20}$ mm.

It will be seen that the plan of jigging sands and slimes together, makes it possible to treat very much finer material with success than has heretofore been supposed possible. The limit for successful work on jigs is generally placed at about 1 mm. The successful jigging of stuff $\frac{1}{8}$ mm. and less, marks a decided advance in the art of dressing. The coarse grains form the interstitial channels* in which this very fine stuff can be concentrated. It is well known that any attempt to treat stuff finer than 1 mm. by itself results in a very imperfect working of the jigs, the losses being large, and the capacity of the jigs small.

The great advantage of this system of jigging is the large proportion of sands successfully treated and finally disposed of by the roughing-jigs alone. Out of 800 tons per day, only 136 tons require further treatment, viz., 30 tons raggings, crushed and treated on the three-sieve jigs, 66 tons fine sand, also treated on three-sieve jigs, and 40 tons of slimes treated on the side-bump tables.

The sole disadvantage lies in the difficulty of forcing all the very finest slimes to go through the jig-sieves. Under normal conditions about 40 per cent. of the stuff below $\frac{1}{20}$ mm. passes off with the tailings. Even this very fine stuff is somewhat impoverished by the action of the jigs, and of that which escapes, a very large proportion is too fine to be saved by any form of apparatus, so that the final loss from this cause is not as serious as might at first appear.

* "Movement of Solid Bodies in Water," "Theory of Jigging," by the writer. See *School of Mines Quarterly*, vol. ix., Nos. 2 and 3. See also the writer's paper, immediately preceding this in the present volume.

The jigs are run with a long stroke, and with as little feed-water and under-water as possible, in order to carry the larger proportion of the fine stuff through the jig-sieve.

It is possible, by lessening the amount of feed-water and under-water, to cause a still larger proportion of fine stuff to pass through the jig-sieve, but at the expense of forcing through, at the same time, much coarser material, and thus sending to the finishing-jigs material that can be treated successfully on the roughing-jigs. The following table shows the effect of reducing the amount of water used in jigging, the percentages of hutchwork and tailings with the normal and with reduced quantities of feed-water and under-water being placed side by side for the different sizes:

	Above ¼ mm.		⅛ to ¼ mm.		$\frac{1}{16}$ to ⅛ mm.		Below $\frac{1}{16}$ mm.	
	Normal.	Reduced	Normal.	Reduced	Normal.	Reduced	Normal.	Reduced
Hutchwork...	26	39	68	94	67	93	60	83
Tailings.....	74	61	32	6	33	7	40	17
	100	100	100	100	100	100	100	100

It will be seen that in each case the maximum percentages of sand pass through the sieve between $\frac{1}{20}$ and ⅛ mm., and between ⅛ and ¼ mm. This indicates that the grain of maximum velocity is about ⅛ mm. in size. This agrees closely with the size called for by theory. The theoretical limit* of successful work on these jigs is therefore about $\frac{1}{16}$ mm.

It would be possible to classify the sands before jigging and to send all this fine stuff directly to the tables. This would necessitate the use of a large volume of water to effect the classification and would involve a large increase in the number of slime-tables.

Experiments are now being made, with every prospect of success, to reduce the loss in this very fine material, without sacrificing the advantages of the present method of treatment.

Spitzkasten.

The hutchwork of the roughing-jigs goes through a series of spitzkasten, respectively 1½ x 7½ feet, 4 x 6½ feet, 8 x 12 feet and 8 x 12 feet in horizontal cross-section, and 2 feet, 7 feet and 10 feet

* *Loc. cit.*

deep. These boxes are built between the posts supporting the main floor of the mill. Their position is indicated in Plates I. and II.

Trunking-Machine.

The material settling in the first box goes to the trunking-machine. This machine consists of a semi-cylindrical iron trough in which revolves a screw conveyer. The trough is divided into two sections, respectively 14 feet and 10 feet in length. The screw conveyer in the lower and longer section is made up of fan-shaped blades of cast-iron, about 4½ inches wide, with a space of about 2 inches between them, breaking the continuity of the screw. The material to be treated flows through rubber hose from the settling-box into this trough; being introduced about midway. The revolution of the shaft forces the galena toward the upper end. The water flows out at the lower end of the trough, carrying with it all the light sand. Wash-water is admitted through a line of spigots at the side of the trough. The upper section of the trough simply serves to convey the galena into the car.

Mineral coarser than ¼ mm. or ⅝ mm. is very perfectly concentrated by this machine and delivered to the car as nearly pure galena. Mineral finer than ¼ mm. is carried off in the tails. The tailings rom this machine are united with the product of the second box of the spitzkasten and go to the three-sieve finishing-jigs.

Finishing-Jigs.

The three-sieve Hartz jigs used as finishing-jigs, as well as for the treatment of the crushed raggings, are of the ordinary side-piston type, and are not figured. They do not require special description. They are run at a speed of 270 strokes per minute; strokes ¼ inch in length. The jigs treating crushed raggings are run slower, 210 strokes, ⅝ inch long, per minute. The jig-sieves are of No. 8 wire cloth (No. 17 wire).

The following table gives the results of typical assays on samples subdivided by sifting:

Size.	Per cent of sand of given size.	Lead in sands treated. Per cent.	Lead in Hutchwork.			Lead in tailings. Per cent.
			1st sieve. Per cent.	2d sieve. Per cent.	3d sieve. Per cent.	
Above ¼ mm.....	8.9	40 99	79.69	70.34	41.80	3.94
¼ mm. to ⅛ mm.....	43.6	16.76				1.02
⅛ mm. to ⅛ mm.....	13.3	16.40				.62
Below ⅛ mm.....	34.2	32.58				11.97
Average.....	100.0	24.75	Average three sieves, 74.00			5.24

The material treated on the finishing-jigs is very rich, containing about 25 per cent. of lead, and the losses are quite large. The material treated is also very fine, over 90 per cent. being less than $\frac{1}{4}$ mm. The losses are confined to the stuff below $\frac{1}{20}$ mm.; above that limit the tailings are poor, with the exception of the stuff above $\frac{1}{4}$ mm., the quantity of which is small (about 3 per cent. of the tails). It will be seen that these jigs treat successfully finer stuff than the roughing-jigs, probably because of the finer jig-cloth and smaller interstitial channels. It is proposed to treat the tailings of these jigs, or at least the finer portions of the tails, on round or side-bump tables.

Parsons-Rittinger Tables.

The Rittinger side-bump table, as modified by Mr. Parsons, is shown in detail on Plate V. The tables are built in pairs, and each table is made double as usual. They are small, each half the double table being about 3 feet by $7\frac{1}{2}$ feet. Instead of being hung from rods the table is supported on four cast-iron feet or "guides," which slide on horizontal steel rods. The latter rest in cast-iron "saddles," bolted to the heavy sill-timbers which run under the whole row of tables. Light spiral steel springs around two of these rods give motion to the table. The tables bump against each other, the blow being taken by a joist of hard wood lying loose between them (not shown in the drawing). The tables are forced apart, against the tension of the springs, by a spiral wedge-shaped cam; 150 bumps, $\frac{5}{8}$ inch long, are given per minute.

The surface of the table is covered with the black enameled duck, sometimes used for desks. This covering is cheap and easily renewed, and furnishes a surface well adapted for the exceedingly fine material treated. The tables are inclined $4\frac{1}{2}°$. There are 16 pairs of double tables in the mill, or 64 tables in all. Of these, 32 are head tables and 32 are used for treating middlings. They are not worked to their full capacity, the head tables treating but $1\frac{1}{4}$ tons each, or $2\frac{1}{2}$ tons to the double tables per twenty-four hours. The material treated is exceedingly fine, 70 per cent. being less than $\frac{1}{20}$ mm.

The following are typical assays of the stuff treated and the resulting tailings. The samples, as before, were subdivided by sifting.

Size.	Percentage of sand of given size.	Percentage of Lead in stuff treated.	Percentage of Lead in tailings.
$\frac{1}{8}$ mm. to $\frac{1}{4}$ mm.....	9.0	2.50	.82
$\frac{1}{20}$ mm. to $\frac{1}{8}$ mm.....	19.4	2.65	.40
Below $\frac{1}{20}$ mm.....	71.6	15.52	2.74
Average.....	100.0	11.75	2.45

Loss in Dressing.

For the year ending May 1st, 1887, the yield of the ore treated was 5.65 per cent. The loss in the tailings is about 2.13 per cent., or 27.4 per cent. of the total amount of lead in the ore. The losses in the Lake Superior copper mills range from 28.5 per cent. to 31 per cent., treating material much more easily saved than this galena. The losses in each case are due to included mineral in the coarser sands, and to finely divided mineral in the very fine slimes. At Bonne Terre an unusually large part of the mineral exists in the ore in an exceedingly fine state of division, as is shown by the large percentage of included mineral in stuff as fine as $\frac{1}{20}$, $\frac{1}{8}$ and $\frac{1}{4}$ mm. This necessitates very fine crushing, and consequently involves the production of a large amount of very fine stuff, much of which contains galena too fine to be saved.

Cost of Dressing.

The cost of dressing for the fiscal year ending May 1st, 1887, was 36.4 cents per ton, divided as follows:

Labor,	13.4
Repairs,	10.0
Supplies,	3.5
Coal,	9.5
	36.4

This compares favorably with the very best Lake Superior practice, the cost of dressing at the Atlantic mill for the last few years being per ton of rock:

	Cents.
1881,	42.54
1882,	37.07
1883,	35.35
1884,	38.95
1885,	30.36
1886,	26.5
1887,	27.5

HATCH
DUMP

ENGINE ROOM

B
STUFFING BOX
H
A
bolts

Plate V.—MUNROE.

ctions of the Ax

JOSEPH LEAD CO.
ARONS' IMPROVED
ERCUSSION TABLES.

Scale of Inches

B. Camshaft.
D. Oil collar Journal box
F. Table
board H. Corner guides
J. Short Saddle
dle L. Spiral Spring
N. Knocker.

FLOOR

The cost at the Atlantic mill (for 1885) can be subdivided as follows:

	Cents.
Labor,	10.86
Fuel,	14.52
Supplies, etc.,	4.98
	30.36

At the Atlantic mill, the water is supplied through a launder above the level of the mill-floor and is not pumped. The ore also is delivered to the mill at a higher level than at Bonne Terre and need not be elevated in the mill. This rock has also been previously crushed by jaw-crushers at the rock-house, the cost of which crushing properly belongs to the cost of dressing, but is not included in the published figures. Finally, the sands are carried off with the water in the waste launders and do not have to be loaded into cars. At least ten cents per ton, therefore, should be added to the Atlantic mill figures in making comparisons of cost at the two places.

The amount of rock treated per ton of coal consumed is as follows:

St. Joseph mill, 1887,	33.59 tons.
Atlantic mill, 1886,	28.34 tons.

The advantage in favor of the St. Joseph mill would be even more striking if an allowance were made for the fuel consumed in elevating water and ore.

The limestone crushed at Bonne Terre is probably not as hard as the amygdaloid rock treated at the Atlantic mill. Somewhat finer screens are used at the Atlantic mill, but the average fineness of the ore is almost exactly the same.

	(1887). St. Joseph mill. (Rolls).	(1886). Atlantic mill. (Stamps).
Above 1 mm.,	43.0	41.6
¼ to 1 mm.,	27.0	27.6
Below ¼ mm.,	30.0	30.8
	100.0	100.0

These figures indicate that it is cheaper to crush by rolls than by steam-stamps; and that the rolls produce quite as large a proportion of slimes as the stamps. The cheapness of the crushing and the large amount of slime, however, are both due in part to the friability of the ore.

It will be difficult to find two mills, either in this country or abroad, treating ore at equally low cost; and the above figures

furnish a powerful argument for the English method of concentrating without size classification.

The losses in the treatment of the very fine stuff are large both at Lake Superior and in Missouri. The remedy in each case will doubtless be found in better methods of crushing and in extending and perfecting the slime-treatment.

A NOTE UPON A MODIFICATION OF THE REDUCING PROCESS USED BY THE CARBON IRON COMPANY.

BY ALFRED E. HUNT, PITTSBURGH, PA.

In a paper written for the Boston meeting of February, 1888 (*Trans.*, xvi., 693), on "Some Recent Improvements in Open-Hearth Steel Practice," the writer described the reducing agent used by the Carbon Iron Company as "a peculiar graphite from Cranston, near Providence, R. I."

Upon an application pending at that time, a patent has since been issued as number 259,795, to Mr. Matthew Graff, for the use of solid carbonaceous matter in the form of coke, charcoal, gas carbon, bituminous or anthracite coal, ground to the size of coarse sand and then coated with a wash of lime, magnesia, clay, or other material of a like character, as an agent to reduce metallic oxides. An amount of this prepared carbonaceous matter equal to from 25 to 30 per cent. of the weight of the ore treated is required to perform the complete reduction. The idea is that the carbonaceous matter is protected against rapid combustion by a suitable coating of some inert material, so that, in effect, its action as a reducing agent will be similar to that of the Cranston graphite previously used, allowing the solid carbon to remain *without combustion* in intimate mixture with the iron oxide at a temperature sufficiently high to reduce the ore and in some cases to melt the matrix in which the iron is held.

In the present practice of the Carbon Iron Company, the use of graphite, except for the formation of bottoms of the reverberatory furnaces in which the reduction is performed, has been entirely abandoned, and "retarded coke," as it is called at the works, has taken its place. Five hundred and fifty pounds of Connellsville coke is ground in a pan to 16-mesh fineness, or is taken in the form of ordinary coke-dust, and made into a hollow basin on the mixing-floor; and seven bucketfuls of a wash composed of equal parts of lime and fire-clay, mixed in water to the consistency of thin cream,

re added. Each bucketful ordinarily contains about one-half pound each of lime and fire-clay. The coating of the carbon is accomplished by mixing with shovels. The furnace-charge of 2240 pounds of ground and moistened ore is laid on the floor in a layer about three inches deep; the "retarded coke" is strewn over the top of it, and the whole mass, having been well mixed with shovels, is then ready for charging. The average yield from 2240 pounds of the ore now used (containing 65 per cent. metallic iron) is as follows, according to its form as sponge, blooms, or muck-bar:

	Pounds.
Sponge, to be melted directly in the open-hearth furnaces,	1550
Squeezed sponge-blooms, of about 6 in. diameter,	1160
Muck-bar, 3½ x ⅝ in.,	1000

The following are average analyses made by the Pittsburgh Testing Laboratory of the squeezed blooms and of the muck-bar produced by the Carbon Iron Company, using "Minnesota Y" ore:

	Squeezed Blooms. Per cent.	Muck-Bar. Per cent.
Metallic Iron,	98.47	99.020
Silica,	0.94	0.560
Phosphorus,	0.020	0.015
Sulphur,	0.021	0.010
Combined carbon,	0.150	0.100

In the open-hearth practice of this company, owing to the present lack of ore-grinding capacity, only 50 per cent. of the sponge-blooms are used, together with 10 per cent. of pit- and shear-scrap and 40 per cent. of pig-metal. With such charges, reckoning the 65 per cent. of iron in the ore added to make the sponge-blooms, and the scrap and pig as being all metallic iron (as is done in ordinary open-hearth loss calculations), the average loss for over six months' running in the two fifteen-ton open-hearth furnaces, now in constant operation, is less than 13 per cent. from ore to steel ingots.

The following are results of tests made upon structural steel plates rolled from the ingots made at these works:

Carbon. Per cent.	Manganese. Per cent.	Elastic Limit. Lbs. per sq. in.	Ultimate Strength. Lbs. per sq. in.	Elongation in 8". Per cent.	Reduction. Per cent.
0.12	0.35	34,000	50,000	30	65
0.14	0.35	36,000	55,000	28	62
0.18	0.35	40,000	60,000	26	56
0.22	0.35	44,000	65,000	25	50
0.25	0.35	48,000	70,000	23	46
0.30	0.35	50,000	75,000	22	42
0.35	0.35	53,000	80,000	20	40
0.40	0.35	56,000	85,000	18	38

A NEW SYSTEM FOR OPERATING REGENERATIVE HOT-BLAST STOVES.

BY JACOB T. WAINWRIGHT, PITTSBURGH, PA.

(New York Meeting, February, 1889.)

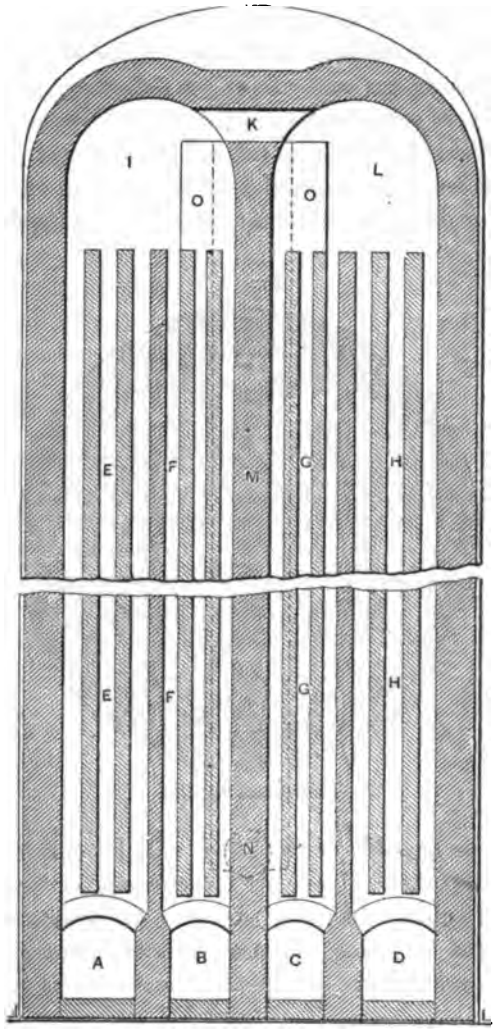
As a means for increasing the efficiency in modern blast-furnaces by supplying to them blast of a much higher temperature than is now possible, the writer offers as a suggestion a modification in the construction of regenerative hot-blast stoves. Stated briefly, the plan proposed is to pre-heat both the gas and the air for its combustion in a manner similar to that generally adopted in Siemens open-hearth furnaces. After thus carrying the temperature of the stove to a point much higher than that common in hot-blast stoves, the cold blast passes through it. Without entering into the details relating to the reversing valves and the system of flues, the general plan may be outlined as follows, referring to the accompanying drawings: The valves are so placed in heating up the stove that the blast-furnace gas enters at B, Fig. 1, and the cold air for combustion at A. Ascending in flues, F and E, respectively, combustible gas and air meet and mix in the combustion-chamber, I. Flowing through K, they descend through the flues, G and H, escaping through the chimney. In this manner the walls of the combustion-chambers, I and L, are highly heated, and the right-hand side of the stove is carried to a higher temperature. By reversing the valves, blast-furnace gas enters at C and cold air for combustion at D. Both are pre-heated in their ascent through the flues, G and H. They burn in the combustion-chamber, L, and the hot gases of combustion, flowing through K, I, E and F to A and B, heat the left-hand side of the stove. In this manner the stove is carried to a temperature greater than that of those now generally in use, in which air and gas are burned without preliminary pre-heating. In the plan proposed, that side of the stove will, of course, be relatively cooler through which cold air and furnace-gas last passed in its ascent to its combustion-chamber.

After having sufficiently heated the stove in the manner outlined, the cold blast is admitted through all the flues, A, B, C and D. It ascends through the highly-heated regenerative material to the chambers at the top and descends through the flue, P, Fig. 2 (dotted

in Fig. 1), through N, Fig. 1, through the open hot-blast valve into the cold-blast main.

It may be urged that in the process of heating up the stove by

FIG. 1.



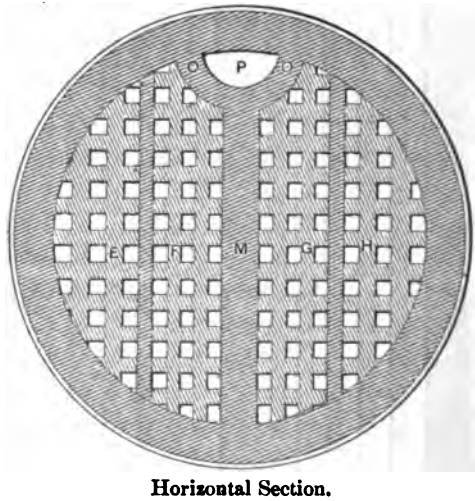
Regenerative Hot-Blast Stove. Vertical Section.

alternately admitting cold air and furnace-gas at a relatively low temperature to one and to the other side of the stove, every operation leaves one side hot and the other side cool. Hence, when the

stove is put in blast, when the cold blast to be heated enters, one-half of it ascends through the cooler side of the stove, while the other half flows upward in the hot side. The two halves unite above, and the whole body of air having only the average temperature of the checker-work of the two sides, is heated by means of the intense temperature of the upper part of the stove.

This objection would have much force if the cold air for combustion and the furnace-gas were allowed to flow through one side of the stove for any very considerable time. The difference in temperature between the two sides of the stove will depend upon the frequency with which reversing is resorted to. If the intervals between the reversals are made short, the temperature of the upper part of the

FIG. 2.



Horizontal Section.

checker-work on one side of the stove will be only slightly lower than that of the other. There are no serious objections to reversing frequently. It does not cause danger; no wear and tear is incurred; and the operating expenses are not increased, because the time of the man employed to operate the valves is not fully occupied.

If it is desirable, however, to carry the temperature of the blast to a point greater than that obtainable by passing it through both sides of the stoves, the cold blast may be cut off from the one side, allowing it to flow only through the hotter side. On the whole, the tendency of practice in Siemens furnaces has been to lessen the

body of the checker-work, and shorten the interval between reversals.

It may be urged, also, that the design submitted does not provide for an adequate mass of material at the top of the stove; that there is no storage-capacity for heat. In reply to this, it may be remarked that aside from the fact that the efficiency of a given quantity of checker-work is dependent upon the frequency of alternations, its storage-capacity for heat increases with the temperature. Material more highly heated will impart heat to the blast more rapidly and yield blast of a higher temperature.

A desirable feature of the design suggested is that the highest temperature is maintained at the top of the stove, so that the material in the lower parts, which supports the greater weight, is thus protected against the effects of excessive heat.

SILICON IN CAST-IRON.

BY W. J. KEEP, DETROIT, MICH.; ANALYTICAL DETERMINATIONS BY H. S. FLEMING, EMPORIUM, PA., AND EDWARD ORTON, JR., VICTORIA, W. VA.

(New York Meeting, February, 1889.)

CAST-IRON, or pig-iron, is iron which contains all the carbon that it could absorb during its reduction in the blast-furnace. As is well-known to chemists, carbon exists in cast-iron in two distinct forms. In chemical union, as "combined" carbon, it cannot be discerned, except as it may increase the whiteness of the fracture, in so-called white iron, or carburetted iron (sp. gr. about 7.72). Carbon mechanically mixed with the iron as graphite is visible, varying in color from gray to black, while the fracture of the iron, according to the color and amount of graphite it contains, ranges from a light to a very dark gray. The specific gravity of graphitic iron is about 7.40.

Silicon in cast-iron is evidently to some extent combined with the iron and the carbon; but whether it exists also in a form corresponding with graphitic carbon, mechanically mixed with the remaining mass, is a question still in dispute, which we have not in this paper endeavored to answer.

It has been the opinion of many writers on metallurgy, and many foundry men, that silicon is in every way injurious to cast-iron. Yet

some brands of siliceous pig-iron have been in high repute for their fitness to mix with other irons and produce soft castings. For many years Scotch pig was deemed, both in America and other countries, indispensable for this purpose. Its manufacture from black-band ores was supposed to impart to it some magic quality. It was discovered years ago, by many American founders, that certain brands of American iron alone could be mixed to make better castings than when Scotch pig was used with them. Moreover, some brands of American iron were found so effectual in softening other pig-irons that they came to be known by the name of "softeners."

The reason for this effect was known to but few. We have before alluded* to the experiments of Prof. Thomas Turner, of Mason College, Birmingham, made to determine systematically the effect of silicon. The result of these researches and of the comments they elicited is, that to-day foundry men understand much better than formerly why they can soften or strengthen their foundry-mixtures by variation of the silicon.

The influence of silicon is indicated to the eye by its action on the carbon which the iron contains. We shall therefore first examine this action.

I. The Effect of Silicon upon the Carbon of Cast-Iron.

Silicon will expel carbon, if the iron, when melted, contains all the carbon that it can hold and a portion of silicon be added. In proof of this we take two grades of Pencost ferro-silicon, as shown in Table I.:

Table I

No. of Test	Ferro-Silicon	Silicon	Carbon	Analysts
491	PENCOST	10.24	1.99	Orton and Lord
493		12.08	1.58	
Increase in Silicon		1.74		
Decrease in Carbon			0.41	
Percentage of increase in Silicon		17.		
Percentage of decrease in Carbon			20.	

* Ferro-Silicon, and the Economy of its Use. Buffalo meeting, October, 1888. Page 253 of the present volume.

Further proof is furnished in analyses collected by Mr. H. M. Howe, and shown in Table II.:

Table II

ANALYST	*a*	*a*	*b*	*b*	*b*	*c*	*d*	*e*	*e*
CARBON	3.33	2.88	2.96	2.42	1.95	1.31	0.79	0	0
SILICON	5.13	5.47	5.92	8.33	9.50	9.80	15.38	20.00	22.00

a, E. Hart; *b*, Zabondsky; *c*, Turner; *d*, Percy; *e*, E. Riley.

These analyses indicate that as silicon increases carbon decreases. The silicons and carbons of the tests of Professor Turner are brought together in Table III.:

Table III

Carbon	1.96	2.00	2.00	2.21	2.13	1.87	2.23	2.01	2.03	1.86	1.81
Silicon	.19	.45	.96	1.37	1.96	2.51	2.96	3.92	4.74	7.33	9.80

These do not show the decrease to be expected on the assumption that additions of silicon will invariably expel carbon. But the increase of the carbons of this series does not prove that in irons *full of carbon* each addition of silicon would not produce a decrease of carbon.

The first iron in the last table is Prof. Turner's artificially-made base-iron, described as prepared by cementing wrought-iron with charcoal. The carbon is only about $\frac{2}{3}$ or $\frac{1}{4}$ of what a regularly furnace-made white iron would contain. This deficiency in carbon would cause the iron to absorb carbon if the conditions favored such absorption, which seems to have been the case.

If the time and heat had been sufficient, and the carbon was present in a condition to be absorbed, the amount in the test with silicon at .45 per cent. should have been nearer 2.5 per cent. of carbon.

If, in Prof. Turner's meltings, the iron remained in the crucible a considerable time, the walls of the crucible may have yielded carbon to it. But if his original metal had contained all the carbon it could hold, it could not have taken any from the crucible. The last number but one of his series, containing 7.33 per cent. of silicon, has not taken up much carbon, and probably contains the amount due to the silicon present. The last number of the series is the ferro-silicon by which he conveyed the silicon to each heat, and is a regularly-made blast-furnace product. Its low carbon is due to its high silicon. We thus see that Table III., while it does not throw any light on the subject of the expulsion of carbon by silicon,

leaves unimpaired the proof of the other analyses presented in Tables I. and II.

Our own analyses (Table I.) show that where carbon has time to escape, as in the blast-furnace, each increase of silicon will drive out a portion of the carbon. In this special case, 17 per cent. increase of silicon decreased the carbon 20 per cent.

We have no proof, so far as we have gone, that an addition to the amount of carbon present will dislodge silicon; on the other hand, our experiments seem to prove that silicon, once in, does not leave the iron on account of variation of heat or of carbon.

The same proposition, regarding the exclusion of carbon by silicon, is true, if we mix different irons and melt them rapidly, without giving the carbon time to escape. If any regularly-made blast-furnace iron is melted rapidly with a portion of ferro-silicon, the melted product will contain a total of too much carbon for the total silicon.

The result must be the expulsion of some of the carbon. By a large number of tests and analyses presented at the October meeting of the Institute in the paper already cited, we proved that the silicon is not expelled. Table I. of this paper proves that when time is given, carbon is expelled. The process by which such exclusion of carbon takes place is worthy of study. The addition of silicon to melted iron holding its full complement of combined carbon decreases its ability to retain that carbon in that state. The instant the carbon is dropped from the combined state it necessarily takes the graphitic form, in which it has only one-third the specific gravity of iron. This causes each particle of graphite to begin an upward movement to reach the surface. As the iron cools, it becomes too thick for the graphite to go further, and at the same time, before it becomes crystalline, its ability to hold carbon constantly decreases, causing still more graphite to be formed. The more graphite, the darker the grain.

II. Description of Our Method of Testing, and Comparisons of Our Tests with those of Prof. Turner.

As in our other investigations, these irons are tested by "Keep's tests,"* the object of which is to make apparent to the eye the effect of various percentages of any element under examination. We

* See the *Iron Age*, vol. xli. Nos. 19, 20 and 22.

have used as bases the same charcoal irons as in other investigations, one, a white iron with 0.186 silicon, and the other a gray iron, "F^{L}M," with 1.249 silicon. To each of these bases we added, in varying percentages, silicon, procured from various ferro-silicons, both American and foreign. We cannot report in this paper all of our experiments, but those which we give illustrate the whole. As already indicated, an objection to the tests of Prof. Turner is, that he did not use cast-iron saturated with combined carbon at blast-furnace temperature. Another objection is, that he introduced, along with his silicon, sufficient manganese to exert a decided influence.

We propose to substantiate the general conclusions of Prof. Turner regarding grain, strength and hardness, by presenting the record of a series of tests almost exactly parallel to his, except that our tests are not subject to these two objections.

Our white iron and his, as shown in Table IV., have almost the same composition, except as to carbon, which is in each case derived from charcoal.

The higher manganese and sulphur in Turner's metal would cause it to be more persistently white than ours.*

Table IV

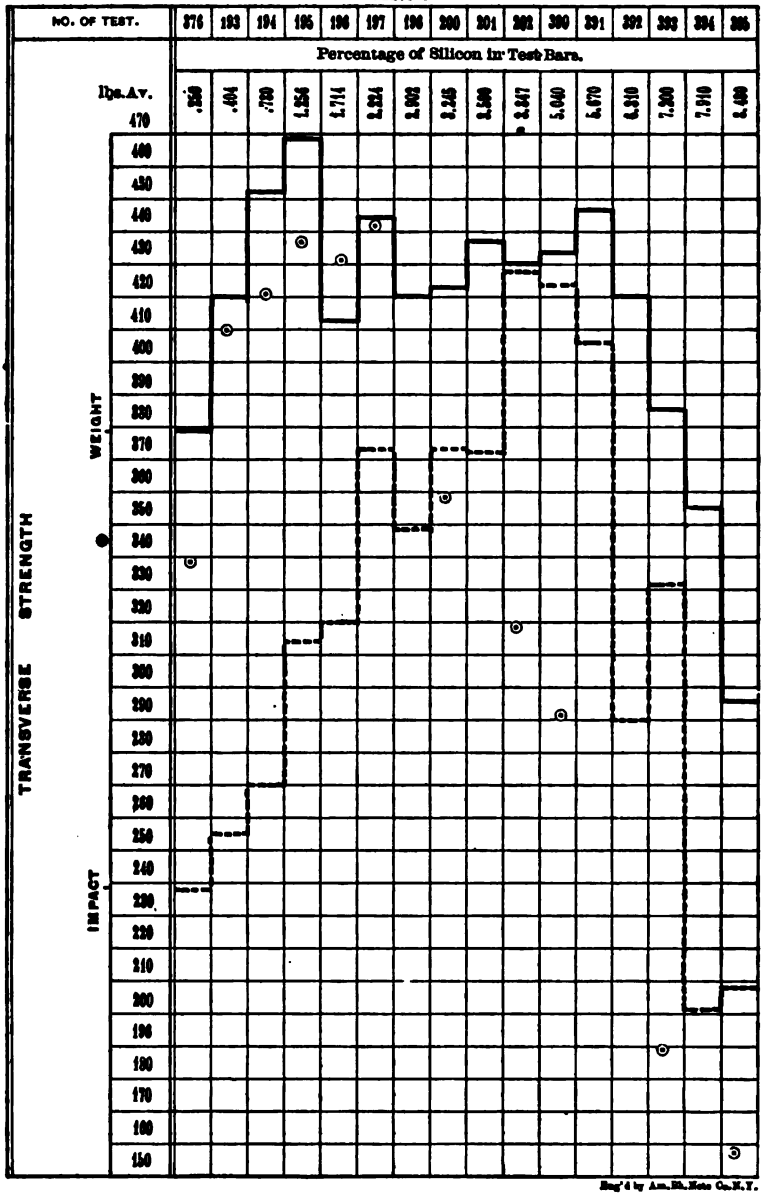
WHITE IRON BASE	Total Carbon	Graphite	Silicon	Phosphorus	Manganese	Sulphur	HOW MADE
Keep's	2.98	.95	.186	.26	.09	.03	Blast Furnace
Turner's	1.96	.88	.190	.32	.14	.05	Cementation
GRAY BASE							
Keep's "F^{L}M"	3.53	3.22	1.249	.084	.187	.04	Blast Furnace

The ferro-silicon by which Prof. Turner introduced silicon contained 1.95 per cent. manganese and 9.80 per cent. silicon. Ours contains only 0.62 per cent. manganese, and 16.32 per cent. silicon. In our tests, therefore, we introduce only .04 per cent., while he introduced .2 per cent. of manganese for each 1 per cent. of silicon. We practically allow all other elements, except silicon, to remain the same in each test, for the reason that, as our ferro-silicon is so high in silicon, we do not use enough of it to vary the phosphorus

* Our gray base, F^{L}M, is almost the exact counterpart of our white base, and also of Turner's, with the exceptions that carbon is still higher, and silicon is sufficient to make the fracture a dark gray. Phosphorus is lower, while manganese is higher, and sulphur is between the two. Both of our furnace-irons are purer than Turner's so far as damaging elements are concerned.

and sulphur or manganese. Table V. exhibits the record of

Table V



strength of this series, and shows the strengths (reduced to a $\frac{1}{2}$ in. sq. area) of Prof. Turner's tests by dots, in the columns nearest cor-

responding to his silicon-contents. The vertical columns contain the records of each individual test, the percentage of silicon actually found in the iron being at the top of the column; also the test-number. The horizontal lines denote measures of transverse breaking-weight in avoirdupois pounds. The figures in the left-hand column show the pounds represented by the horizontal record-line in each vertical column. The full record-line represents the resistance to a weight gradually applied at the center of a bar 1 foot long and $\frac{1}{2}$ inch square. The dots are Turner's calculated transverse breaking-weights for his tests reduced to our size of bar. The broken line records the stress which ruptured our bar by impact. This is recorded, for comparison, in pounds, found by calculating the foot-pounds for each fall and using an arbitrary value in pounds* for a foot-pound. The record in all cases is either an average of two or of three test-bars.

In determining impact breaking-weight, the first blow is always struck with a fall of $\frac{1}{4}$ inch, each succeeding blow increasing by $\frac{1}{8}$ inch fall, until fracture takes place. The blow is delivered by a hammer of 25 pounds, swinging on a 6-foot radius. Both the transverse and impact machines are constructed specially for these tests, and each makes an autographic diagram. Prof. Turner concludes from his tests that the amount of silicon producing the maximum increase of strength is about 1.80 per cent. Our tests show that this conclusion is correct.

But this is only true when a white base is used, for the following reasons: A white iron, of the composition of our white base, will not produce sound castings. The blow-holes account for its lower specific gravity, and for its diminished strength. A small addition of silicon, from .80 to 1.80 per cent., eliminates blow-holes and causes sound castings.

If an iron is used as a base which will produce a sound casting to begin with, each addition of silicon will decrease strength, as we shall show hereafter.

III. The Influence of Silicon upon the Strength of Cast-Iron.

Our conclusions regarding the effect of silicon in percentages less than 1.50 per cent. are drawn from the records in Table V., above.

* The value assumed as a basis of calculation, for reasons not necessary to be given here, is 288 pounds avoirdupois for a fall of 25 pounds through $2\frac{1}{8}$ in.

As soon as the casting is sound, with the least graphite, the greatest crushing-strength is obtained. This condition gives also the greatest specific gravity. Any further addition begins to form graphite, relieves the metal of its brittleness, and gives at a certain point the greatest tensile and transverse strength. When the graphite increases beyond this point, the metal is divided by it, the strength decreasing as the graphite increases. At the same time the metal grows softer. To decide as to the action of silicon in amounts greater than 1.50 per cent., we have made a large number of tests.

As they are in the main similar, we shall present the four series of tests mentioned in our paper on Ferro-silicons, before referred to, and numbered from 340 to 355 and also a series of tests with a uniform $2\frac{1}{2}$ percentage of silicon; these latter, and one-half of the former, being with the same $F^{L}M$ base. (The silicons in these $2\frac{1}{2}$ per cent. tests were calculated with such care that a chemical determination was not deemed necessary.) As foundry castings contain between $1\frac{1}{2}$ and 3 per cent. of silicon, and for thin castings nearer $2\frac{1}{2}$ per cent., these series show the effect of silicon in ordinary practical foundry-work, when the percentage of silicon is produced artifically by mixtures. We show also a series of pig-irons containing from $1\frac{1}{2}$ to 3 per cent. of silicon, in which the percentages of silicon are produced naturally in the blast-furnace.

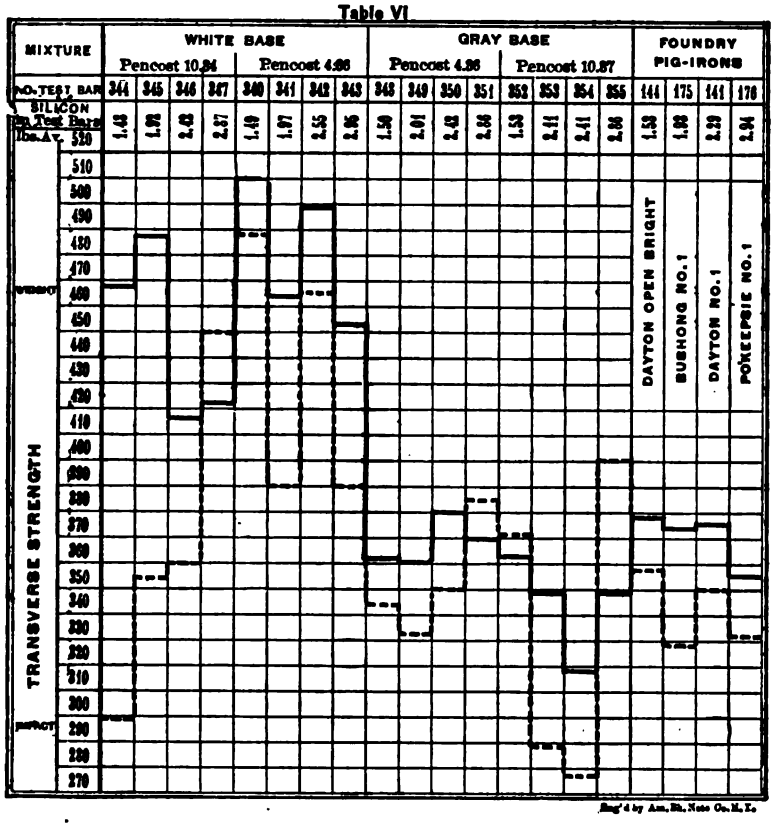
Pure iron, if it could be made, would be so pliable and inelastic as to possess but little strength. Simple iron, holding all the carbon that it could take up, would be white, hard and brittle, because the carbon would all be combined, and such iron could not be made into a solid casting.

As we have said, silicon changes the carbon from the combined to the graphitic state. Cast-iron which contains enough silicon to take out the brittleness, and to allow it to make a solid casting, is the strongest composition that we ordinarily find in natural cast-iron.

It will be noticed in any test that we have made, that a *solid* casting having its carbon combined, is stronger than one in which the carbon is more graphitic. For strength, therefore, we must endeavor to obtain, instead of a perfectly uniform distribution of graphite, a concentration in uniformly distributed minute pockets, around which the iron holding combined carbon may form a lace-work. If strength be more important than softness, we will leave the greatest possible quantity of carbon in the combined state that can be left, without causing the iron to be brittle.

The percentage of silicon required depends greatly upon the condition in which the carbon exists in the iron to begin with. For example, the presence of 2½ per cent. of silicon in an iron will not determine the relative quantity of the two kinds of carbon. If the pigs contain enough silicon (as in F^{L}M, which has 1½ per cent.) to produce considerable graphite, then the addition of 1½ per cent. of silicon (bringing the total up to 2½ per cent.) will turn a large amount of the remaining carbon into graphite; but, if into the

Table VI



white base which contains only ⅛ of one per cent. silicon, we add 2⅜ to make the silicon 2½, we shall find that the carbon persistently holds to the combined form, and that the graphite which is formed, forms pockets, where it acts like a cushion to remove brittleness.

The tests in Table VI. are arranged to illustrate these statements. Tests 344 to 347 contain the largest amount of combined carbon of all the five series in this table. These irons were made from the

white base, which made porous and brittle castings when used alone, and to which was added a ferro-silicon, containing less carbon than the base, and all of it graphitic. The silicon is in a concentrated form, and the ferro-silicon used alone, makes a very weak and brittle casting. The casting from this mixture contains less carbon than the white base, and this decrease diminishes the power of the silicon to form graphite. These castings are therefore brittle and hard, while the first two are white. The increased strength of test 345 is due to the casting being more solid.

Series 340 to 343 shows silicon added to the white base in a diluted state, and a resultant increase of carbon, for the reason that the ferro-silicon contained more carbon than the base, though a large portion was combined. As the casting cooled, therefore, the contained silicon caused more of the carbon to take the graphitic form than in the series last spoken of, and removed brittleness at the same time, leaving enough combined carbon to give very great strength.

In series 348 to 351, the gray base $F^{L}M$ is used, which contains much more carbon than the white base. In this series the ferro-silicon containing most combined carbon is used.

The large amount of graphite in the mixture divides the mass and weakens the casting as compared with the series from the white base; but the strength is greater than in the next or fourth series, where a ferro-silicon was used containing practically no combined carbon. In this series everything tends to increase graphite and consequently makes weak castings. The most interesting series is the last in the table, showing the strength of pig-irons with nearly the same percentages of silicon as the previous series. The perfect mixture of the elements of the iron causes these natural irons to be stronger than the artificially-mixed irons last considered. When in a later part of this paper we examine the grain of the members of these series, we shall see even more clearly the advantages of using pig-irons which have the proper proportion of silicon, rather than bringing up that proportion by mixture.

From an examination of this table we conclude that if the silicon is taken up by the pig in the furnace, a smaller amount will produce a given quantity of graphite than when the silicon is added at the re-melting.

We also conclude that in a pig that has taken up enough silicon in the furnace to turn a portion of its carbon to graphite, the combined carbon remaining will turn into graphite more readily than in a pig which has its carbon mostly in the combined state, by reason of a lack of silicon.

Table VII. further proves what we have stated by a series of casts, all from the gray base $F^{L}M$, the silicon being added by ferro silicons, which contain varying percentages of silicon and varying proportions of combined carbon.

This series proves that variations in the percentage of silicon will not insure a given strength or physical structure, but that these results will depend upon the physical properties of the original irons.

Table VII

Test of Fe.Si.		36	397	403	140	214	398	213	221	85	89	396	401
Name of FERRO-Si. used with $F^{L}M$ (Si 1.946)		Wellston	Pencost	Pencost	Dayton	Govan	Pencost	Foreign	Foreign	Wellston	Wellston	Foreign	Pencost
SILICON In Ferro Silicon		5.66	4.36	12.08	4.35	9.85	6.78	11.89	10.62	6.99	5.15	16.32	10.34
NO. TEST		447	350	448	449	450	451	452	453	454	455	456	354
TRANSVERSE STRENGTH	WEIGHT 380												
	370												
	IMPACT 360												
	350	$F^{L}M$											
	340												
	330												
	320												
	310												
	300		Si. 2½ PER CENT.										
	290		in each Test										
	280												
	270												
Fe-Si	WEIGHT	301	478	150	360	195	486	626	161	340	366	144	280
	IMPACT	198	466	76	390	180	363	277	91	394	226	68	135

To bring up the silicon to 2½ per cent. with $F^{L}M$ as a base, only 1¼ per cent. of silicon has to be added; and as this takes but a small portion of a 10 or 12 per cent. metal, the condition of the carbon in the ferro-silicon is not so important as its condition in the base.

A very important question is, Which is best, a ferro-silicon carrying 10 per cent. and more of silicon, or one with about 5 per cent.?

The tests made thus far, and our practical experience, are rather in favor of the low percentages, probably because with the low-silicon irons more of the iron must be used, and as there is more carbon in low than in high grades, more carbon will be introduced; also because the proportion of carbon in the ferro-silicon and the base will be more nearly the same. We have recorded in Table VII. in column 447, the strength of the base F[L]M, the weight above and the impact below the 350-pound line, and at the bottom of each column the strength of each ferro-silicon used. The strength of the resultant castings is not in any way dependent upon the strength of the irons from which they are made, but can be traced to the state of the carbon in the original irons. Thus far we have considered the question as though we expected to account for all variations by the chemical constitution of the iron, but this is by no means the case. There is evidently a mechanical or physical condition of structure not dependent upon the chemical composition. We have often pro-

Table VIII

STRENGTH OF PENCOST								
Silicon in Test Bars		4.35	6.75	8.07	9.27	10.29	10.83	11.63
NUMBER OF TEST		397	398	399	400	401	402	403
1st. Remelt	Weight	473	338	262	216	231	221	150
	Impact	425	254	125	142	125	125	76
2nd. Remelt	Weight	433	270	200	219	239	235	197
	Impact	417	266	130	107	130	119	101
NUMBER OF TEST		405	406	407	408	409	410	411
Silicon in Test Bars		4.25	6.57	7.99	9.31	10.24	10.76	11.48

duced castings with one part strong, soft, and gray, and another part of the same casting white, hard, and brittle, a drawing of which we show further on. Many of the bars in the tables just considered show such structure. Why in one case the carbon took the combined state, and in the other the graphitic, we cannot say.

Fast or slow melting or cooling allow these physical changes to occur. Less or more heat, or any other change of conditions, may cause this physical change. For this reason chemical analyses do not lead to explanations of these peculiarities as readily as properly conducted physical tests. The founder cannot be certain what a mixture of irons will produce. Experience will guide him, but only physical tests will show exact results. For example, it would be supposed that with silicon at 2½ per cent., and no appreciable variation in other elements, the resultant irons would be alike; but Table VII. shows how great a divergence exists.

After enough silicon has been added to cause solid castings, any further addition and consequent increase of graphite weakens the casting. The softness and strength given to castings by a suitable addition of silicon is, by a further increase of silicon, changed to stiffness, brittleness, and weakness.

These and other tests show also that silicon by itself is a weakening agent. To prove this we produce in Table VIII. the tests of a number of ferro-silicons, all made at one furnace under similar conditions; as silicon increases the irons grow weaker.

All our tests have proved that after silicon has reached 4 or 5 per cent., all the graphite has formed that will form, and that the weakening effect of silicon is more apparent beyond this percentage.

IV. The Influence of Silicon upon Deflection.

While Table V. shows that the strength increased up to 1½ per cent. of silicon, Table IX. does not exhibit much corresponding increase of deflection.

Up to this point in this series, graphite has not formed in sufficient amount to be perceptible to the eye. From this point on, the deflection increases (but the strength decreases) as graphite increases.

Table IX

NO. OF TEST		376	193	194	195	196	197	198	200	201	202	390	391	392	393	394	395
		Percentage of Silicon in Test Bars.															
		.250	.404	.780	1.256	1.714	2.224	2.902	3.245	3.580	3.847	5.040	5.670	6.210	7.200	7.940	8.480
DEFLECTION. Inches	.30																
	.20																
	.10																

We say, therefore, that as strength decreases from increase of graphite and decrease of combined carbon, deflection increases, or, in other

Table X

DEFLECTION OF PENCOST	Silicon in Test Bars		4.35	6.75	8.07	9.27	10.29	10.83	11.68
	1st. Remelt	No. Test	397	398	399	400	401	402	403
		Bars	.23	.18	.13	.12	.12	.10	.09
	2nd. Remelt	Bars	.22	.15	.12	.11	.11	.10	.09
		No. Test	405	406	407	408	409	410	411
	Silicon in Test Bars		4.25	6.57	7.99	9.31	10.24	10.76	11.43

words, bending is increased by graphite. When no more graphite can form and silicon still increases, deflection diminishes, showing that high silicon not only weakens iron, but makes it stiff. This stiffness is not the same strength-stiffness which we found at the beginning of the series, caused by compact iron and combined carbon. Table X. illustrates the *brittle-stiffness* of silicon.

Table XI

MIXTURE	WHITE BASE								GRAY BASE								FOUNDRY PIG-IRON			
	Pencost 10.36				Pencost 4.36				Pencost 4.36				Pencost 10.36							
NO. TEST	344	345	346	347	340	341	342	343	348	349	350	351	352	353	354	355	144	175	141	176
SILICON in Test Bar	1.48	1.92	2.42	2.87	1.49	1.97	2.56	2.76	1.50	2.04	2.42	2.80	1.53	2.11	2.41	2.86	1.53	1.92	2.29	2.94
Inches																				
DEFLECTION ACTUAL .30 .20 0																				
DEFLECTION RATIO .0009 .0007 .0006 .0005																				
																	DAYTON G.B.	BUSHONG NO.1	DAYTON NO.1	POTTSVILLE NO.1

In Tables XI. and XII. we have plotted the ratio of the deflection to the weight; but although this is more exact, yet the total deflection is so nearly the same that we find it is not necessary to calculate the ratio.

It will be seen that deflection is proportional to the graphite. We introduce Table XII. of deflections to show further that deflection does not vary proportionately with strength. This series contains 2½ per cent. of silicon in each test.

Table XII

Test of Fe Si	36	397	403	140	214	398	213	221	25	39	396	407
Name of FERRO-SI used with FLM (Sl.1.249)	Wellston	Pencost	Pencost	Dayton	Govan	Pencost	Foreign	Foreign	Wellston	Wellston	Foreign	Pencost
SILICON in Ferro-Silicon	5.66	4.86	12.06	4.35	8.85	6.76	11.99	10.62	6.99	5.15	10.82	10.94
NO. TEST	447	350	448	449	450	451	452	453	454	455	456	364
DEFLECTION ACTUAL .30 .25 .20												
DEFLECTION RATIO .0008 .0006 .0004												

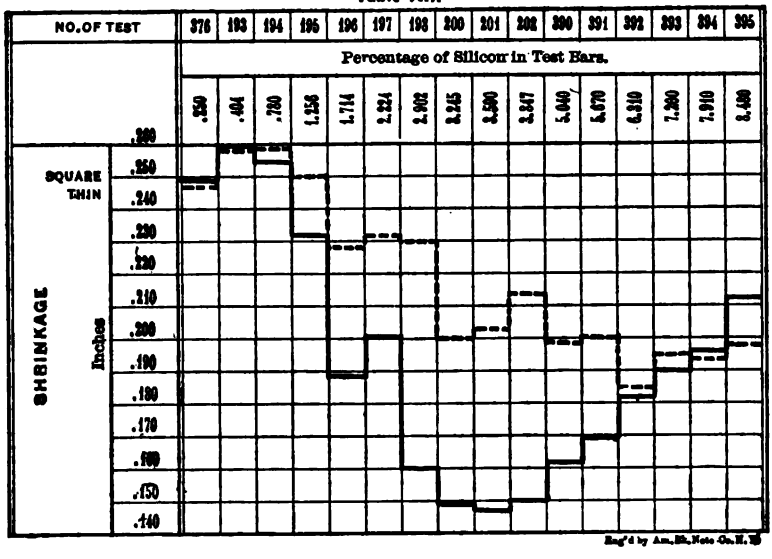
In Table XI. we have shown the deflections of impact by a broken line.

The permanent set varies nearly as the deflection, being greatest in the iron that contains most graphite. The iron that has the greatest deflection, probably, though not necessarily, has the greatest set. Many cast-irons take no set, and are, therefore, perfectly elastic. The subject of the elasticity of cast-iron will be treated by itself at another time.

V. The Influence of Silicon upon Shrinkage.

We do not yet thoroughly understand all the influences which

Table XIII



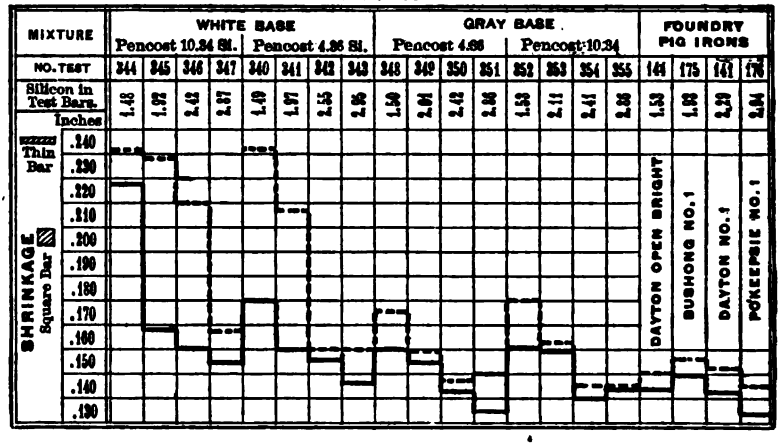
cause cast-iron to shrink in solidifying and cooling. The simple loss of heat from the fluid metal will not account for it. If this were the case the shrinkage would indicate which iron requires the highest temperature for melting, which is not the case.

Starting with the assumption that pure iron would shrink a certain amount for each reduction in temperature during cooling, we will assume, also, that if the molten iron contains any element which, when the iron becomes crystalline, will crystallize out, and in so doing will swell and still remain mechanically mixed with the iron, then such a casting will show less shrinkage than a casting of pure iron, and less than one in which the contained element remains in combination, instead of crystallizing out.

Several elements probably act in this way, and thus reduce shrinkage. Carbon certainly does. As we have seen, it is caused by silicon to crystallize out as graphite. The greatest formation of graphite seems to be just before the iron becomes solid. The volume of graphite is about three times as great as that of iron. The graphite cannot swell to its full extent on account of the mass and incipient crystallization of the iron about it. It causes a swell at the instant when the iron crystallizes, but not enough to counteract all shrinkage of the metal. This explanation will account for the popular belief that gray iron expands the instant it becomes crystalline. If the swelling of the graphite causes a swell *before* the iron begins to shrink, the iron may shrink as much as if the graphite were not present, yet the shrinkage would be lessened by exactly the amount of the swell.

Table XIII. shows that in the first three heats there was not enough graphite liberated to produce any decrease in shrinkage, but it is seen that when graphite does separate shrinkage decreases. Table XIV. shows this even better. The tables show the shrinkage of our test bars one foot long.

Table XIV



Another thing worthy of notice is, that in pig-irons which received their silicon while in the blast-furnace, the graphite more easily separates, and the shrinkage is less than in any mixture. It will be noticed in each table that all the evidence is in favor of using irons that contain the proper amount of silicon for the use to which they are put.

To show that shrinkage is not determined simply by the amount of silicon present, we introduce Table XV.:

Table XV

TEST OF FE-SI	36	397	403	140	214	398	213	221	85	89	396	401
Name of FERRO-SI used with FL-M	Wellston	Pencost	Pencost	Dayton	Govan	Pencost	Foreign	Foreign	Wellston	Wellston	Foreign	Pencost
SILICON in Ferro Silicon	5.66	7.36	12.68	4.35	8.85	6.76	11.90	10.62	6.99	5.15	16.32	10.24
NO. TEST.	447	350	448	449	450	451	452	453	454	455	456	354
SHRINKAGE SQUARE. THIN Inches .200												
.190												
.180												
.170	FL M											
.160												
.150												
.140												
.130												
Fe-Si	.166	.139	.170	.139	.218	.178	.187	.196	.176	.155	.209	.200

Engd & by Am. Bk. Note Co. N.Y.

We shall find in future examinations that other elements than graphite act in reducing shrinkage.

Silicon, indeed, operates by itself to increase shrinkage. This is indicated by Table XVI.:

Table XVI

SHRINKAGE OF PENCOST	Silicon in Test-Bars		4.35	6.75	8.07	9.27	10.29	10.83	11.68
	NUMBER OF TEST		397	398	399	400	401	402	403
	1st. Remelt	Thin Bar	.146	.173	.187	.194	.201	.181	.173
		Square Bar	.130	.177	.194	.196	.199	.175	.171
	2nd. Remelt	Thin Bar	.175	.171	.185	.193	.202	.181	.177
		Square Bar	.136	.178	.190	.192	.200	.176	.170
	NUMBER OF TEST		405	406	407	408	409	410	411
	Silicon in Test-Bars		4.25	6.57	7.99	9.31	10.24	10.76	11.48

It will be noticed in this table that as silicon increases, shrinkage also increases.

The same increase is also seen in Table XIII. In the foreign

ferro-silicon used in forming this series, the shrinkage was .311 with Si 16.32. Thus far the greatest shrinkage we have found in iron, white from lack of silicon, was .270. The uniformly increasing shrinkage, in series with increasing silicon, and the exceptionally high shrinkages of ferro-silicons, and also the rise in shrinkage in white irons to which silicon has been added, but not in sufficient quantity to cause graphite to be seen, as in the first three numbers of Table XIII., seem to prove that silicon of itself increases shrinkage, though by reason of its action upon the carbon in ordinary practice it is truly said that silicon "takes the shrinkage out of cast-iron."

The slower a casting crystallizes, the greater will be the quantity of graphite formed within it. We think, however, that there is a double action with silicon, and that a large amount of carbon is turned into graphite at the instant when crystallization takes place. This accounts for very thin castings being gray, and for the chill being lessened. If our records are examined it will be found that with reasonably high silicon, say from 5 to 7 per cent., the thin bar of our tests shrinks as little or less than the square bar. This shows that time (that is, slow cooling) does not, in all cases at least, continue to increase graphite. We have expressed the opinion that silicon of itself shrinks much more than iron, and thus we account for the high shrinkages of silicon-irons. This may account also for the square bar shrinking more than the thin bar; but we shall venture another opinion in addition to this: Silicon seems sometimes to form a compound with combined carbon, and in so doing prevents the further formation of graphite. In another part of this paper, discussing the grain of the iron, this action will be made apparent. In foundry-practice, when a mixture of irons is made to bring the silicon up to the desired percentage, castings will occasionally be produced having the entire surface gray to the depth of $\frac{1}{32}$ to $\frac{1}{4}$ inch, while the center of the casting is perfectly white, with so much shrinkage as to be full of minute cracks. Analyses of these parts of the casting show substantially the same chemical constitution. Fig. 1 is a drawing of such a casting. A sample of this iron was analyzed by Mr. Eliot A. Kebler, with the following results:

	White inside. Per cent.	Gray outside. Per cent.
Graphite,	1.307	1.874
Combined carbon,	2.554	1.754
Silicon,	2.742	2.846
Manganese,	.501	Not est.

It is evident that the total carbon was high in the casting as a

whole. The gray surface had cooled rapidly, and enough of the carbon had changed to graphite to form a light-colored gray grain. The white central portion is so sharply divided from the gray that it can be split away from it. It contains a little more carbon than the gray part, about one-third of which is graphitic, while in the gray, one-half is graphitic. In both cases the combined carbon is high. Silicon is practically the same in both parts, and other analyses show manganese to be the same. Still another analysis showed that sulphur and phosphorus were practically alike in both. Why, then, did the central part, which had the longest time to cool, turn white, with $2\frac{3}{4}$ per cent. of silicon, if not for the reason that the silicon and combined carbon united to form this intensely white compound, which was so compact that it could cover up the $\frac{1}{3}$ graphite? This leads to the interesting question, which we will not here pause to discuss, How can so much graphite remain invisible?

FIG. 1.

Such an explanation of the action of silicon and carbon will account for the great shrinkage and also the hardness of high-silicon irons. As an example of the same thing, Mr. Orton made a ferro-silicon containing over 9 per cent. of silicon, which was a true mottled iron, showing much white. The sample of Wellston white iron (test 42), which we have mentioned in former papers, contained 2 per cent. of silicon.

Irons low in silicon and high in combined carbon have more shrinkage in the thin than in the square bar, and the thin bar is the whiter. When the excessive shrinkage is from combined silicon, the shrinkage of the thin bar will be as little or even less than that of the square bar, and the thin bar will be darkest.

An iron that can hold a large amount of silicon and not tend to form compounds of silicon and carbon, and does not liberate all the graphite at the instant of crystallization, but liberates graphite a little later, will shrink less than any other iron, because the continued formation of graphite more than counteracts the shrinkage due to the silicon. We have found irons of this kind which contained as much as 5 per cent. of silicon and which shrank as little in our bars as .120 inch. Such irons have a very fine and compact grain, and make strong castings.

VI. The Influence of Silicon upon Hardness.

The general subject of the hardness of metals has been ably presented in a paper by Prof. Turner,* in which he not only gives the history of the methods pursued by others in determining hardness, but describes his own sclerometer and gives directions for its use.

For determinations of hardness we employ this apparatus, having procured from Prof. Turner the diamond, and a standard scratch made upon a strip of platinum, that we might make our results conform to his standard. We have proved the accuracy of our work by having more than fifty specimens numbered, and, without knowing what the irons were, determining the hardness, and then arranging the results in the graphic form of Table XVII.

The correspondence of these results, with what we know regarding the mechanical properties of the irons, was proof that the machine produced a correct record. The results shown in this table will be a surprise to many, and will greatly modify the reputation that silicon has attained as a softener.

Silicon has been considered a softener when added to hard cast-iron. Is this a fact?

Silicon changes carbon from the combined form, which is hard, into graphite, which is soft. Graphite in cast-iron divides the mass and thus allows the tool to cut the intervening thin webs of metal. From 2 to 5 per cent. of silicon will change into graphite about all of the combined carbon that can be so changed. This change, we know from experience, reduces hardness. The series of irons of which the hardness is shown in Table XVII. are the same which are considered elsewhere in this paper, and also in the paper on ferro-silicon, read at the Buffalo meeting of the Institute, October, 1888.† The two series at the top, and the two at the bottom of Table XVII., exhibit the decrease of hardness brought about by the change of carbon to graphite.

We shall soon prove that silicon of itself, however small the quantity present, hardens cast-iron; but the decrease of hardness from the change of the combined carbon to graphite is so much more rapid than the hardening produced by the increase of silicon,

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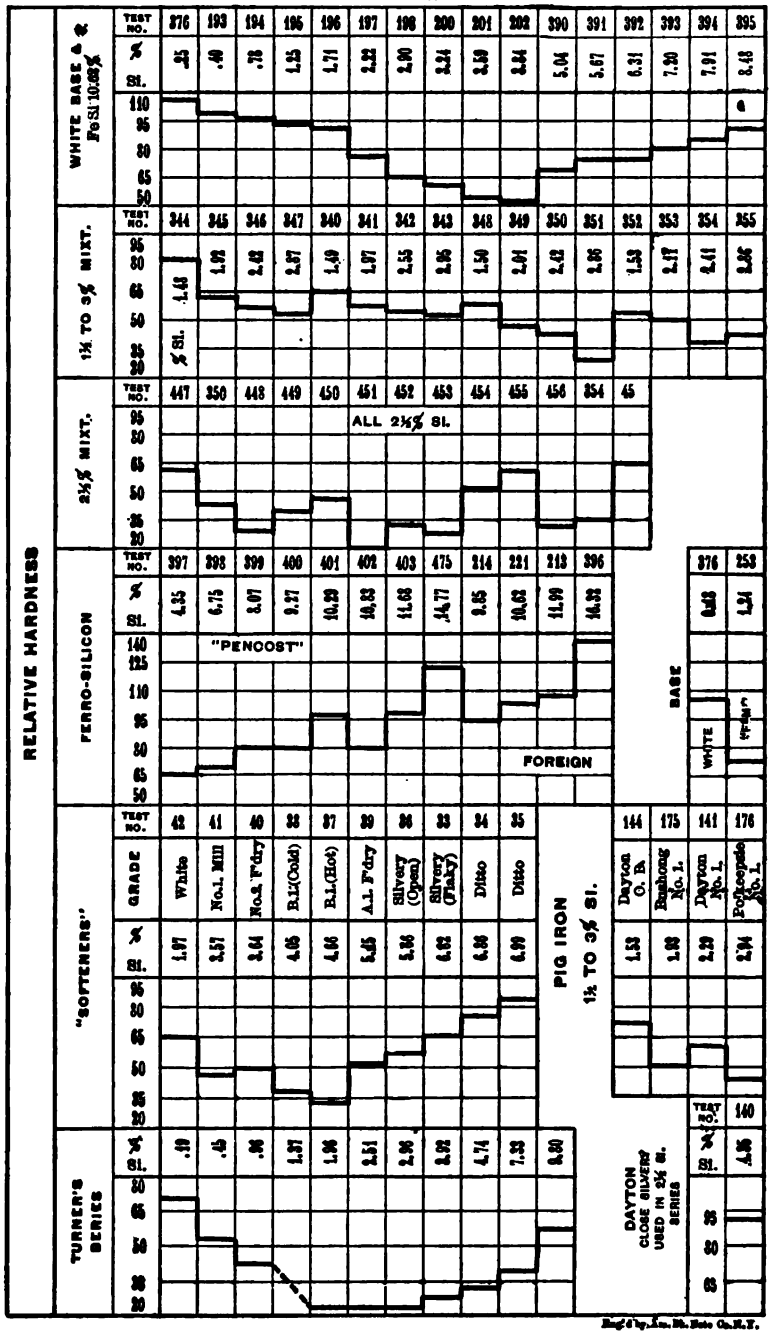
* *Proceedings of the Birmingham Philosophical Society*, vol. vi., Part II.

† For diagrams showing other physical characteristics of the irons called "softeners" in Table XVII., consult the chart in a paper describing these tests, published in the *Iron Age*, May 31st, 1888, p. 886, and there find also a drawing of our Turner machine.

Table XVII


RELATIVE HARDNESS
TURNER'S SERIES
"SOFTENERS"
FERRO-SILICON
2½% MIXT.
1¾ TO 3% MIXT.
WHITE BASE & Fe Si 10.65%
"PENCOST"
ALL 2½% SI.
FOREIGN
BASE
WHITE
MOTTLED
PIG IRON
1¾ TO 3% SI.
DAYTON CLOSE SILVERS USED IN 2¾ SI. SERIES
TEST NO.
% SI.
GRADE
White
No.1. Mill
No.2. F'dry
B.I.(Cold)
B.I.(Hot)
A.I. F'dry
Silvery (Open)
Silvery (Flaky)
Ditto
Ditto
Dayton O. B.
Buchanan No. 1.
Dayton No. 1.
Pottsgrove No. 1.

that the total effect is to decrease hardness, until the silicon reaches from 2 to 5 per cent.

As practical foundry-work does not call for more than 3 per cent. of silicon, the ordinary use of silicon does reduce the hardness of castings; but this is produced through its influence on the carbon, and not its direct influence on the iron.

We see in all the series that, after the bulk of the carbon has become graphitic, any further addition of silicon hardens cast-iron. If graphite is still increasing, the increase is so slow that the total effect is nevertheless one of hardening, and this hardening is so uniformly apparent as to convince us that it is the direct effect of silicon. In each series hard irons appear at each extremity; but the cause of the hardness at the beginning is combined carbon; at the end, excessive silicon.

The hardness of white iron deficient in silicon, such as our white base (test 376), which has a hardness of 105, is due to combined carbon. Prof. Turner's white base had a hardness of 72, also produced by combined carbon. Probably the reason that the hardness in the latter is less than in 376, is that the iron contained less combined carbon, as well as total carbon, than 376. We infer from this and from other tests that the greater the percentage of combined carbon in a cast-iron the greater the hardness.

The changing of the combined into graphitic carbon diminishes hardness more rapidly than anything else. In each series we see that when this change has ceased to diminish hardness, say at from 2 to 5 per cent. of silicon, the hardening by the silicon itself becomes more and more apparent as the silicon increases. That this increase of hardness is not due to a recombination of the carbon is proved by the analyses of the irons produced by Prof. Turner, and is proved more conclusively by the analyses of "Pencoet" 10 and 12 per cent. ferro-silicons (401 and 403). In the 10.34 per cent. metal, the combined carbon is only 0.07 and yet the hardness is 95. In the 12.08 per cent. metal, the combined carbon is only 0.06 and yet the hardness is slightly more. In the 14 per cent. iron, combined carbon is still low, and yet hardness is 120. The same is true of the 16 per cent. metal, with hardness 135. This is the hardest iron we have found.

We now come to a most important part of this subject. After constructing Table XVII., it was natural to expect that hardness would correspond with some of the other characteristics of the same irons, as exhibited in previous tables. We expected a similarity

between hardness and chill, and were surprised to find no connection between the two. Strength and hardness are not enough alike to point to a common cause; that is, we cannot say that a weak iron is always hard and a strong iron always soft.

But when we compared shrinkage and hardness we found them almost exactly proportional, and we feel safe in saying that when silicon varies, *and other elements do not vary materially*, castings with low shrinkage are soft; and that as shrinkage increases, the castings grow hard, in almost, if not exactly, the same proportion. For ordinary foundry-practice we can therefore make the scale of shrinkage also the scale of hardness, provided variations in sulphur, and phosphorus especially, are not present to complicate the result.

The importance of this discovery is at once apparent. Without specially preparing a casting, its shrinkage can be measured with the greatest accuracy by a person with little experience in testing, as soon as it is cold; while great experience is required to test the hardness of cast-iron by any other known method. For a Turner machine, the most satisfactory of all methods, the specimen must be very carefully prepared. The observation must be made in a very careful manner, under peculiar conditions, and the eye and judgment of the operator play an important part in the result. Measuring of shrinkage is simple, positive and correct.

We shall, in future papers, show that low shrinkage does not always indicate that an iron is soft. For general every-day practice, however, shrinkage seems to be the best scale for hardness. We do not refer to what is called "more or less" shrinkage from mere inspection of the pig, but to actual measurement of a test-bar made from the pig.

VII. The Influence of Silicon upon the Chilling Quality.

Molten cast-iron in contact with a body that rapidly abstracts its heat, becomes white or "chills" at and near the surface of contact. This chill varies in depth with a certain quality of the iron. Not all irons will "chill." The term "chilling irons" is generally applied to such as, cooled slowly, would be gray, but cooled thus suddenly assume the character, either to a depth sufficient for practical utilization (*e.g.*, in car-wheels) or only so far as to be detrimental. Many irons chill more or less in contact with the cold surface of the mould in which they are cast, especially if they are thin. Some-

times this is a valuable quality, but for general foundry purposes it is desirable to have all parts of a casting an even gray.

Silicon exerts a very powerful influence upon this property of irons, partially or entirely removing their capacity of chilling. Examining Table XVIII. we see that the chill does not decrease, except in intensity, until the silicon reaches 2 per cent. At each addition the chill changes from a lamellar zinc-chill at first to a finer and more silky surface, until in the 2 per cent. bars the gray

Table XVIII

NO. OF TEST		376	193	194	195	196	197	198	200	201	202	390	391	392	393	394	395
		Percentage of Silicon in Test Bars.															
		.250	.404	.780	1.254	1.714	1.224	2.902	3.245	3.590	3.847	4.040	4.670	4.810	7.200	7.940	8.480
CHILL Inches	.80	All	All														
	.60																
	.40																
	.20																
	.00																

is sufficient to permit the depth of chill to be measured. The chill decreases as silicon increases, until as the silicon becomes very high the combination of silicon and combined carbon that we have before spoken of (if such a combination is the explanation) forms a new chill.

Table XIX

MIXTURE	WHITE BASE								GRAY BASE								FOUNDRY PIG IRON			
	Pencost 10.84				Pencost 4.86				Pencost 4.86				Pencost 10.84							
NO. TEST	344	345	346	347	340	341	342	343	348	349	350	351	352	353	354	355	144	175	141	176
SILICON In Test Bar	1.48	1.92	2.42	2.87	1.49	1.97	2.56	2.96	1.50	2.01	2.42	2.86	1.53	2.11	2.41	2.86	1.53	1.88	2.29	2.84
CHILL Inches .80	All	1.25			1.25												D. O. B.	B. No. 1.	D. No. 1.	P. No. 1.
.60																				
.40																				
.20																				
.00																				

Table XIX. shows the chills of the four series and the pig-irons previously examined. We see that as silicon increases the chill decreases. This is, however, not alone the effect of the percentage of silicon present, but equally of the state of the carbon in the iron to begin with. This consideration accounts for the lack of a regular

decrease in the pig-irons. The chill in these pig-irons depends more on some influence during the smelting, which caused the proportioning of the combined and graphitic carbon, than upon any difference in chemical constitution.

In the above table the gray base alone has a chill about $\frac{3}{4}$ inch deep. As it contained $1\frac{1}{4}$ per cent. of silicon, the further addition of another equal amount brings the chill down to less than $\frac{1}{2}$ inch.

To show more clearly that it is not the percentage of silicon alone, but also the peculiar state of the carbon, that determines the depth of chill, we introduce Table XX., representing the chills of the $2\frac{1}{2}$ per cent. silicon series used before. If the removal of chill is the thing desired, there is reason for a careful selection of the ferro-silicon by which the silicon is added. In all these tests the F^{L}M base with a $\frac{3}{4}$-inch chill was used.

Table XX

Test of Fe Si	36	397	403	140	214	398	213	221	35	39	396	401
Name of FERRO-SI used with F L M	Wellston	Pencost	Pencost	Dayton	Govan	Pencost	Foreign	Foreign	Wellston	Wellston	Foreign	Pencost
SILICON in Ferro Silicon	5.56	4.96	12.06	4.35	9.35	6.76	11.99	30.62	6.99	6.15	14.32	10.34
NO. TEST	447	350	448	449	450	451	452	453	454	455	456	354
CHILL Inches .30												
.20												
.10												
.00												

If we were to use for foundry-castings irons taking a deep chill we should not be able to take out all the chill by the addition of silicon, as this would take more than 3 per cent. Si, which has already been shown to be too high for a foundry-mixture. Therefore, to obtain castings without chill the initial irons should have the least possible amount of chill. Again, silicon cannot be depended upon to give unvarying results. The tendency of the carbon is to hold persistently to the state in which it exists in the pig. A certain percentage of silicon in the casting or in the pig does not by any means insure an iron of a given grain, or of a given depth of chill.

It is so absolutely impossible to know the peculiar disposition of the carbon in pig-iron bought in the open market, and therefore to be able to judge how much silicon will produce a given effect, that the only satisfactory way is to purchase pig-iron that, used alone, will give as nearly as possible the desired qualities to the castings. It is then an easy matter to tone up the iron with a little silicon, if necessary. As a general rule high-silicon irons do not show any chill when melted alone.

VIII. The Influence of Silicon upon Fluidity.

Under this head we can do little more than produce the records of fluidity of the irons already considered.

These records are a measure in inches of the length that a strip $\frac{6}{100}$ inch thick will run when poured from one end lying horizontal. The mould is 12 inches long. We cannot as yet think of any better test. If the temperature of the melted iron is less at one time than another, the fluidity for that cast is less. In judging of the fluidity of any given brand of iron we take the average of a large number of tests. The result in Table XXI. seems to prove that when sili-

Table. XXI

NO. OF TEST	376	193	194	195	196	197	198	200	201	202	390	391	392	393	394	395
	Percentage of Silicon in Test Bars.															
	.250	.404	.780	1.256	1.714	2.124	2.902	3.245	3.580	3.847	4.040	4.670	6.310	7.200	7.910	8.400
FLUIDITY Inches 8																
6																
4																
2																
0																

con is mixed with irons previously low in silicon the fluidity is increased. Table XXII. proves the same thing, and brings out

Table XXII

MIXTURE	WHITE BASE								GRAY BASE								FOUNDRY PIG IRON			
	Pencost 10.84				Pencost 4.96				Pencost 4.96				Pencost 10.84							
NO. TEST	344	345	346	347	340	341	342	343	348	349	350	351	352	353	354	355	144	175	141	176
SILICON in Test Bar	1.48	1.92	2.42	2.87	1.49	1.97	2.55	2.96	1.50	2.01	2.42	2.86	1.53	2.11	2.41	2.93	1.53	1.93	2.29	2.94
FLUIDITY Inches 10																				
5																	D.Ch.	E.No. 1.	Chill.	Chill.
0																				

another point, viz.: The fluidity is somewhat dependent upon the amount of graphite there will be in the final casting.

On account of the large amount of graphite introduced with the 4 per cent. Pencost, though the percentage of silicon is the same as in the first series, we see that the fluidity is slightly greater. The two F^{L}M series contain much more carbon and more graphite than the white series, and, as we would expect, we find the fluidity

Table XXIII

Name of FERRO SILICON used with FLM.	Wellston	Pencost	Pencost	Dayton	Govan	Pencost	Foreign	Foreign	Wellston	Wellston	Foreign	Pencost
SILICON in Ferro Silicon	10.00	4.36	12.00	4.36	9.05	9.76	11.90	10.00	9.90	5.15	10.02	10.34
NO. TEST.	447	350	448	449	450	451	452	453	454	455	456	364
FLUIDITY. Inches 10												
5												
0												

greater. The graphite in the 4 per cent., being less than in the 10 per cent. ferro-silicon, the fluidity of the fourth series should be, and is, greater than that of the third. We have before remarked that silicon introduced while the iron is in the blast-furnace acted more forcibly than when artificially introduced. In this table the series of pig-irons show this to be true.

Table XXIII. shows that it is not the percentage of silicon, but the state of the carbon and the action of silicon through other elements, which causes the iron to be fluid. Of this proposition Table XXIV. presents further proof, showing that too much silicon may cause the iron to be thickly liquid. This table gives results from Pencost iron.

Table XXIV

FLUIDITY OF PENCOST								
Silicon in Test-Bar		4.35	6.75	8.07	9.27	10.29	10.83	11.63
1st. Remelt	NO. TEST	397	398	399	400	401	402	403
	Bars	9.75	9.00	4.50	7.50	7.37	7.75	——
2nd. Remelt	Bars	10.62	8.37	7.37	9.50	9.12	8.62	8.00
	NO. TEST	405	406	407	408	409	410	411
Silicon in Test-Bar		4.25	6.57	7.90	9.31	10.24	10.76	11.43

Silicon-irons have always had the reputation of imparting fluidity to other irons. This comes, no doubt, from the fact that up to 3 or

4 per cent. they increase the quantity of graphite in the resulting casting.

IX. The Influence of Silicon upon Grain.

Under this head we employ the fracture of the chilled end of the square test-bar, as *a*, Fig. 2, to show the grain of the chill, as well as the appearance of the transition from chill to gray; also, the fracture of a thin strip, *b*, with a rib on its under-side which is a very sensitive indication to the eye of any slight variation in the quality of the metal; also the fracture of the thin bar, *c*, which was cast with the square bar. (Of this and the ribbed bar the accompanying figures show only one-half of the bar; but the two edges of the thin bar are hardly ever alike.) We also show the central cross-fracture of the square bar *d*.

The fracture of these sections indicates to the eye the effects which have been described in previous parts of this paper, and shows the

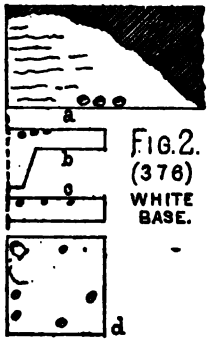


Fig. 2. (376) WHITE BASE.

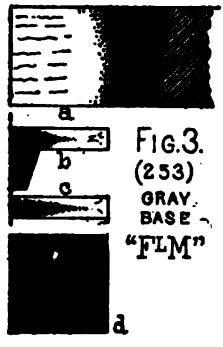


Fig. 3. (253) GRAY BASE "FLM"

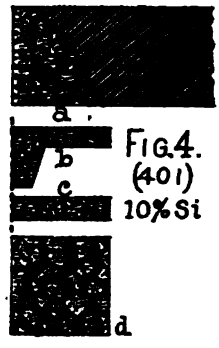


Fig. 4. (401) 10% Si

influence of the silicon upon the carbon; and of the silicon itself in the change of color and in the altered character of the crystallization.

Fig. 2 shows the perfectly white fractures of our white base, and also the blow-holes almost always present when the metal is melted alone. (The corner *d* was lowest in the mould.) Fig. 3 shows the fracture of F^{L}M, gray base. The chill, *a*, is from $\frac{1}{2}$ to $\frac{3}{4}$ inch deep. The edges of both the ribbed strip, *b*, and thin strip, *c*, are white, the center being a very light gray. Observe that the gray is surrounded by the white, indicating that the coldness of the mould caused the whiteness, and that the separation of graphite at the center was on account of the center being hot longest. The cross-fracture, *d*, is in this case darker at the corners and in the center, showing most combined carbon between these portions. Quite often in this iron this is reversed, the center being light and the corners having a slight chill.

Often with this base the strips, *b* and *c*, are almost, or wholly, white. The chill is like zinc-crystallization, only more silky. Irregular white and dark patches mix where the chill ends. In our tests it is the change in appearance of the fractures of these two base-irons of Figs. 2 and 3 that shows the influence of silicon.

It is extremely difficult to show the grain of a ferro-silicon. Fig. 4 represents reasonably well the fracture of 10 per cent. Pencost (401). Other ferro-silicons would be more or less flaky, or more or less white with still a flake, or a black surface covered with shining dust, or with flakes so small as to look very little like flakes. Ferro-silicons generally show no chill, but carry the same grain to the chill-surface, and the same grain in the thin as in the thick castings.

To illustrate the change produced by silicon we will give the appearance of the fractures of the bars in the series in the tables with numbers 340 to 343, where silicon runs from 1½ to 3 per cent. In

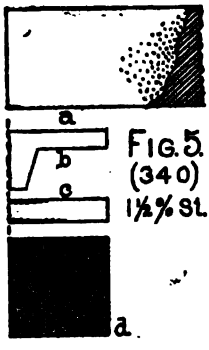


FIG. 5. (340) 1½% Si.

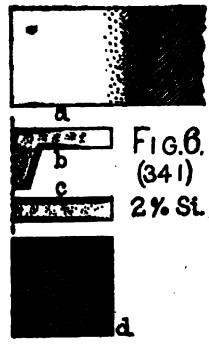


FIG. 6. (341) 2% Si.

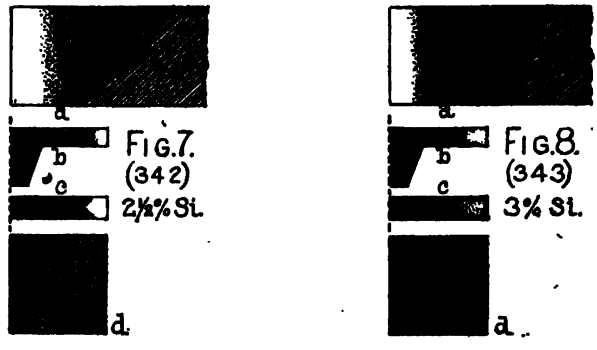
this series the white base (Fig. 2), and the ferro-silicon containing 4.36 Si were used. Fig. 5 shows the grain after silicon has reached 1½ per cent. (340). The chill runs 1¼ inches, though black grains begin to show at $\frac{6}{10}$ inch. The strip, *b*, is tin-white, as is also the thin strip, *c*, with the exception that near the center of the bar the surface is a light gray. The square bar is composed of gray and white blotches, irregular in shape.

Fig. 6 is the 2 per cent. member of this series. The chill is nearly flat. The change from white to gray is sharper, and the depth is only one-half as deep as the last. The strip, *b*, has the surface gray and gray spots appear near the center. The thin bar, *c*, has its flat surfaces gray, while the center is white with a faint mixture of gray.

The square fracture is a perfect mottle, the white lines being dim.

These white net-work lines are more prominent at the surface of the casting. Fig. 7 is a representation of the 2½ per cent. member of this series. The chill is only ⅛ as deep as the last, but more flat and white, and the division between white and gray is still sharper. The bar, *b*, and the thin strip, *c*, have a gray center with a light grain at surface, the white increasing so that at the edges the bar is white. The gray centers are a beautiful mottle. The square fracture is still darker than the 2 per cent. fracture, and the white thread lines of the mottle are more indistinct.

Fig. 8 is the 3 per cent. member of this series. The bars, *b* and *c*, are gray mottle, but the white at the edges is more prominent than in the last set, also instead of white surfaces this set has black soft surfaces, showing more graphite at the surface than the 2½ series. This shifting of the graphite from one part of the casting to another is a peculiarity of silicon-iron when mixed with other irons. To

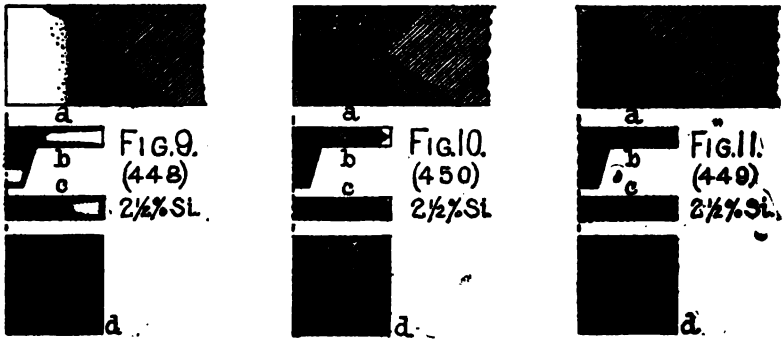


illustrate further this last-mentioned tendency of silicon, drawings of a few of the 2½ silicon series will be shown. A drawing of each member of this series would be interesting, as no two are alike; but the whole may be grouped under three general kinds of fracture.

Fig. 9, which is the fracture of 448, fairly represents also 452, 453 and 454. The chill is white, fine and silky, and ⅛ of an inch deep. The strip, *b*, is a common fracture in silicon mixtures where only two irons are used. The surfaces are a jet black, and the center of the rib intensely black, with silver dust appearance. The nose of the rib is often brilliantly white and in this case is bounded by a jet black surface. The thin strip, *c*, will bear the same description, except that all bounding surfaces are black leaving white cores near the edges. The square fracture is a very dark spongy grain, darker at corners. This set shows the deposition of graphite on the

cooling surfaces, and the uniting of silicon with combined carbon in the center (if this is the correct explanation). These intensely black surfaces would indicate to the eye the softest black grain, and that the silver dust might be brushed off; but the scratch of a sharp point brings out a pure silver-white line, showing that the silver grains are the pure metal shining through the thin black coating. The metal if polished is very white and the grain very dense. The peculiar colors and the variation in the depth and appearance of chill in these 2½ per cent. tests, show the effect of 2½ per cent. of silicon, only either the silicon or the carbon, or both, in each ferro-silicon, is in a different state, and therefore produces different effects.

Fig. 10 shows the fracture of test 450, and represents fairly 447, 451, 455 and 456, although, as we said before, the proportion and shape of dark and light may vary. This ferro-silicon was "Govan," and we suppose it is understood that the base in all of these 2½ per cent.



mixtures is F^{L}M. The chill is almost entirely wanting, black grains running clear to the surface. In such a case we average the chill, and in this case call it $\frac{4}{100}$ of an inch. Back of this chill the grain is intensely black, more like lampblack. Further back the true grain appears. The strip, *b*, has intensely white edges. The grain back of these edges is like lampblack, and the grain is also as black on the nose of the rib. Inside these dark parts is a brighter and more open grain, but at the center is a sharply-defined black spot. The thin strip, *c*, has an even open grain with black edges and surfaces. The square fracture is a perfect dark sponge, darker at the corners and edges. The last of the 2½ silicon series that we shall illustrate is the one where Dayton silicon silvery iron is mixed with F^{L}M. It is shown in Fig. 11 (449). We introduce this because it lacks the peculiarities exhibited in all the other figures. The chill

is nearly nothing, yet is a true chill; we average it at $\frac{6}{100}$ inch. Back of the chill for ½ inch the grain is a small and distinct mottle, with each white boundary complete. Back of this the grain is a dark sponge. The strips, *b* and *c*, are a similar mottle, only the white lines are more distinct than in the chill fracture. The square fracture is a perfectly even, dark, spongy grain. This set of fractures shows the most perfect grain of any member of the series, though a 2½ per cent. cast with 14 per cent. "Pencost" ferro-silicon is about as perfect.

We intended to represent the grain of one or all of the four pig-irons shown in the tables from 1½ to 3 per cent. silicon; but the fractures would show exactly like Fig. 11, except that the depth of chill varies. Both 144 and 141 have an open spongy grain, while 175 has an indistinct dark mottle, and 176 a larger and more distinct mottle. In all these pig-irons there is no variation in the grain in *b*, *c*, or the square bar, except that in 144 there is a slight chill on the edge of *b*, and this same strip in 175 has a ¼-inch chill on each thin edge. Tests 141 and 176 are even gray throughout. The peculiar chilling-quality due to silicon is apparent in all of the tables. The perfection of the grain of pig-irons, and the absence of the peculiarities observable in mixtures, leads us to say again that it is more desirable to obtain the proper proportion of silicon in the pig-iron to be used than to seek to produce it by subsequent mixing.

X. General Remarks.

From the statement in Prof. Turner's papers, that the maximum strength occurs with just such a percentage of silicon, and his statement that a founder can, with silicon, produce just the quality of iron that he may need; and from his naming the composition of what he calls a typical foundry-iron, some founders unacquainted with the action of silicon, have inferred that if they knew the percentages of silicon in their irons and in their ferro-silicon, they need only mix so as to get 2 per cent. of silicon in order to obtain, always and with certainty, the maximum strength.

We have shown that the solution of the problem is not so simple. Each of the irons which the founder uses will have peculiar tendencies, given them in the blast-furnace, which will exert their influence in the most unexpected ways.

A review of Table VI. will show how wide is the range of strength in the five 2 per cent. silicon-irons shown, due to the peculiar state of the carbon in these irons when they came from the fur-

nace. There is no typical composition for cast-iron made by mixture; in the table referred to, the chemical composition of all the 2 per cent. irons is essentially the same, and yet the tests show the greatest diversity among them physically.

We have seen, however, that a white iron which will invariably give porous and brittle castings can be made solid and strong by the addition of silicon; that a further addition of silicon will turn the iron gray; and that as the grayness increases the iron will grow weaker; that excessive silicon will again lighten the grain and cause a hard and brittle as well as a very weak iron; that the only softening and shrinkage-lessening influence of silicon is exerted during the time when graphite is being produced, and that silicon of itself is not a softener, or a lessener of shrinkage, but through its influence on carbon, and only during a certain stage, it does produce these effects.

PROMINENT SOURCES OF IRON-ORE SUPPLY.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(New York Meeting, February, 1889; first announced at the Birmingham meeting, May, 1888, but completed with later figures.)

THE estimated product of iron-ore in 1888 throughout the world was, in round numbers, 50,000,000 gross tons, of which the United States produced about one-fourth. Great Britain leads this country in production by about 15 per cent., but the output of the United States is practically equal to that of Spain, France, Russia, Austria and Hungary combined. We produced last year nearly twice as much iron-ore as Spain, with all her exports; and we mined 20 per cent. more than Germany and Luxemburg. In addition to the domestic iron-ores consumed in this country, 587,470 gross tons of foreign ore was imported, and the amount of mill-cinder used in blast-furnaces is estimated at 450,000 gross tons. The total quantity of iron-ore required in 1888 for blast-furnaces, rolling-mills and other purposes in the United States was approximately 12,700,000 gross tons. The purpose of this paper is to indicate the principal sources of this supply, and call attention to the extent of mining operations covering many years in certain localities, which, although some of them now produce but a small proportion of the above total, have in the aggregate added very largely to the material

wealth of the country, and have been of great service in the development of our iron industries. The actual figures have been given where they are reliable; but for some of the older mines complete records are unattainable, and in such cases estimates have been secured from the best available authorities. Besides presenting the details, the general features have been grouped for more convenient comparison in the accompanying graphic chart, from which an idea of the development from year to year, and the present condition of the various sources of supply, can be obtained.

I. The Lake Superior Region.

This is by far the most productive iron-ore region in the United States. The output for 1888 was 5,023,279 gross tons, equivalent to about 40 per cent. of all the iron-ores mined in the country. The total output since the beginning of operations to the close of 1888 is very close to 41,000,000 tons, divided as follows:

Total Product of the Lake Superior Region.

Range.	First opened in	Product gross tons.	Proportion per cent.
Marquette	1854	27,130,419	66.5
Menominee	1877	8,594,037	21.0
Gogebic	1884	3,586,876	8.8
Vermilion	1884	1,498,209	3.7
		40,809,541*	100.0

The work of the past year is exhibited in the following statements:

Iron-Ore Product of the Lake Superior Region in 1888.

Range.	No. of Mines Listed.	No. Active during the Year.	Product. Gross Tons.	Increase or Decrease over 1887. Gross Tons.	Proportion in 1888. Per cent.
Marquette	74	41	1,921,525	61,482 inc.	38.3
Menominee	36	22	1,165,039	34,304 dec.	23.2
Gogebic	34	21	1,424,762	139,497 inc.	28.4
Vermilion	2	2	511,953	117,701 inc.	10.1
Total	146	86	5,023,279	284,376 inc.	100.0

* To this may be added 2320 gross tons classed as miscellaneous in the statistical tables.

The product of each range, except the Menominee, was the largest in its history.

Range of Product of Single Mines in 1888.

Range.	Number of Mines each producing in Gross Tons,			
	over 400,000	over 200,000	100,000 to 200,000	50,000 to 100,000
Marquette	.....	3	3	5
Menominee	.....	1	4	4
Gogebic	1	1	2	2
Vermilion	1	.....	.....	1
Total	2	5	9	12

The Marquette Range.—On this range, the Lake Superior mine leads with a total, from its opening in 1858 to 1888 inclusive, of 4,435,356 tons, the output of 1888 being 240,225 tons. For nine consecutive years this mine has produced over 200,000 tops annually. In 1887 it shipped 302,909 tons.

The Cleveland, which ranks next, has produced since its opening in 1854 to 1888 inclusive, 3,889,150 tons. Its product, however, of 184,316 tons in 1888 was less than that of any year since 1879. The maximum, in 1884, was 225,674 tons.

The Republic stands third in aggregate product, having mined 3,038,514 tons since 1872, including 235,064 tons in 1888. This output has been exceeded by the mine five times since 1879, the maximum being 277,757 tons in 1884.

The Jackson was opened in 1856, and, like the Cleveland and Lake Superior, is considered one of the older mines. Its aggregate to date is 2,816,366 tons, and its maximum output was 132,297 tons in 1871. It produced 101,909 gross tons in 1888.

The Champion was opened in 1868 and reached its maximum in 1884, when 210,180 tons were mined. It has contributed 2,082,574 tons to the total Lake Superior output, of which 174,680 tons was taken out in 1888.

The Lake Angeline, opened in 1864, had produced up to the close of 1880 about 500,000 tons. After 1883 the product was largely augmented and in 1888 reached 223,600 tons, the aggregate to date being 1,311,706 tons.

The above list includes every iron-ore mine in the Marquette range

which has produced over a million tons, except the York (or New York), the aggregate of which is 1,040,129 tons, but which has been practically inactive since 1883. The Barnum, Humboldt, Michigamme, Salisbury and Winthrop mines, however, have yielded over half a million tons each.

Among the heavy producers during 1888 may be mentioned the Cambria (57,865 tons), the Cliff's Shaft (78,520 tons) and the Pittsburgh and Lake Superior (56,321 tons).

The Menominee Range.—The general features in the history of the leading mines on this range to the end of 1888 are tabulated as follows:

Product of Leading Mines in the Menominee Range.

Mine.	No. of Years' Operation.	Aggregate Product. Gross Tons.	Maximum Annual Product. Year.	Maximum Annual Product. Gross Tons.	Product in 1888. Gross Tons.
Chapin	9	1,938,117	1887	336,128	272,278
Vulcan	12	1,150,882	1887	205,036	129,541
Norway	11	1,112,735	1880	198,165	87,260
Dunn	*a*	138,925	1888	114,248	114,248
Florence	*b.*	545,226	1882	160,155	142,586
Iron River	7	509,370	1888	110,000	110,000
Commonwealth	9	492,043	1882	115,862	61,818
Ludington	9	630,323	1885	124,194	61,883
Mastodon	7	193,873	1888	51,463	51,463
Perkins	10	375,474 *c*	1883	76,514	16,684

a. Opened in 1887. *b.* Opened in 1880. Worked intermittently in recent years. *c.* This is one of the mines formerly ranked as heavy producers, but now of less importance. Of the total given, 40 per cent. was mined in 1882 and 1883.

The Gogebic Range.—Here there are six mines of leading importance as producers. The first 1000 tons of iron-ore shipped from this range in 1884 was from the Colby, which has now shipped a total of 887,154 tons, all of which, except 1022 tons, was mined in four years. The mine reached its maximum (285,880 tons) in 1888.

The Norrie ranks next to the Colby, having in four years supplied an aggregate of 769,713 tons. This mine, however, took first

rank in 1888 with a yield of 412,196 tons, and the question as to whether the Norrie or the Minnesota Iron Company's operation is to be considered the greatest producer in the Lake Superior region depends upon whether a number of openings are considered as one mine or not.

The Aurora also reached its maximum output in 1888, viz., 179,937 tons, bringing its total up to 437,991 tons.

The Ashland, with a product for 1888 of 164,134 tons, was more than 11,000 tons below its maximum output in 1887, but it has in four years produced an aggregate of 420,181 tons.

The Germania was also 7800 tons below its output for 1887, producing in 1888 53,918 tons, and making its total for four years 140,834 tons.

The Iron King has produced in three years 164,058 tons, of which 60,218 tons was mined in 1888, and 74,656 tons in 1887.

The Kakagon yielded in 1887 52,179 tons, and in 1888 but 1228 tons; and the Puritan, which furnished 45,000 tons in 1887, was inactive in 1888. Although the Bessemer, Ironton, Nimikon, Odanah, Sunday Lake and Superior, which gave promise in 1887 of heavy shipments, were idle in 1888, yet the Gogebic range maintained its position next to the Marquette as a producer.

The Vermilion Range.—This range, the fourth in order of prominence of the mining districts of the Lake Superior region, has in five years contributed 1,498,209 tons, of which 511,953 tons were supplied in 1888. Of this amount 457,341 tons came from the openings of the Minnesota Iron Company, and represent the maximum output of that organization. The remaining 54,612 tons came from the Chandler mine, as its first contribution to the ore-supply of the country. The product of this mine was somewhat restricted by the absence of shipping facilities until late in the season.

In my paper on the "Resources of the Lake Superior Region" (*Trans.*, xvi., 168) will be found statistics somewhat similar to those given above, but covering the period to the close of 1886 only. A comparison of the two statements will emphasize some very important changes that have occurred. The same paper contained a number of ore-analyses, and a graphic representation of the yearly product of Lake Superior region as compared with the Bilbao district of Spain; also a table showing the annual product of the Lake Superior and Bilbao districts from 1860 to 1886, inclusive.

The great quantity of ore handled in the Lake Superior region is the more remarkable when its high percentage of iron is considered.

The greater portion of the ores are red hematites, varying from the soft to the hard specular varieties; but some magnetites are obtained, and in a few deposits the hydrated condition of the ore places some of it close to or within the limits of the brown hematites.

The distance from these mines to blast-furnaces, and the expense of transportation makes it necessary to maintain in shipments a standard of quality above the requirements of most other iron-producing districts. This standard may be considered as approximating a yield of 60 per cent. of iron when dried at 212° F., but many of the ores exceed this requirement.

II. The Cornwall Ore-Banks.

Notwithstanding the magnificent record of the Lake Superior region, no single mine or operation under one management has approached in output the Cornwall ore-banks in Pennsylvania, the yield of which in 1888 was 722,921 gross tons. This is the largest amount ever taken in one year from the Cornwall deposit, which has been operated for 150 years, and has produced an aggregate of 9,395,922 gross tons of iron-ore to the close of 1888.

The development of this property has naturally been stimulated by the advance in iron-production. The product had not amounted to a half-million tons in any year until 1885; and 1879 was the first year in which over a quarter of a million tons was mined. The total product of the past decade has been 4,421,000 tons, which is nearly one-half of the aggregate output from the beginning.*

The ease with which the soft magnetites are mined from the Cornwall ore-hills, and the convenience of the deposit to fuel-supply and to blast-furnaces, would permit a still greater output if efforts were made to distribute the sales of ore over a larger area than is now the case. It is, however, equitable in comparing the Cornwall ore-hills with the Lake Superior region to note the fact that ores which yield about the same percentage of iron as the Cornwall ore are carried to the waste-piles at many of the Lake Superior mines, and that an output of 575,000 to 600,000 tons of the average Lake Superior ore, as shipped, would produce as much iron as the 723,000 tons of the Cornwall product.

The sulphur and copper in the Cornwall ore do not interfere with its use in producing Bessemer iron. The former is generally reduced

* Yearly statistics, from 1854 to 1885, inclusive, and a general description of this remarkable deposit, appear in Mr. d'Invilliers' paper on "The Cornwall Iron-Ore Mines, Lebanon County, Pa.," *Trans.* xiv., 873.

by roasting with solid fuel in the Gjers and Grittinger kilns. Lately the Davis-Colby kiln has been supplied with waste gases from the blast-furnace as fuel. The percentage of copper is not sufficient to affect materially the use of the ore.

III. The Lake Champlain Region.

Another important supply of iron-ore comes from the deposits of magnetite in the Adirondack or Lake Champlain region. These ores differ from the Cornwall ore in being hard and generally free from sulphur, which must be removed from Cornwall ore by roasting. Many of the Lake Champlain ores now mined are of Bessemer quality, but some carry a considerable percentage of phosphorus and are used for foundry-iron. These ores are also employed as fettling or fix in puddling-furnaces.

The ores shipped carry from 55 to 68 per cent. of iron, but occasional masses of ore of practical purity are obtained in mining. A sample lately taken from a pile of about 400 tons at one of the New Bed openings of Messrs. Witherbees, Sherman & Co., analyzed by Mr. T. R. Woodbridge, shows,

Iron,	72.00
Phosphorus,	0.018

and daily analyses made by Mr. Woodbridge of ore from this opening show that in 8000 tons removed within three months the maximum and minimum percentages were as follows:

Iron,	69.20 to 72.04
Phosphorus,	0.011 to 0.026

The Adirondack Mountain region has many large deposits of magnetites, which are unwrought or worked on but a limited scale, because, in some instances, the distance from fuel prevents utilizing the lean ores, while in others the ores carry titanium in such quantity as to interfere with their development.

The production of the Lake Champlain district for the last ten years is as follows:

Years.	Gross tons.
1879,	420,341
1880,	531,000
1881,	637,000
1882,	675,000
1883,	500,000
1884,	504,894
1885,	379,077
1886,	588,752
1887,	768,852
1888,	789,419
Total,	5,794,335

Some of the Lake Champlain deposits have been operated for nearly a century. The total output of this district to date is estimated at 15,147,000 gross tons, of which 5,794,000 tons have been mined within the last decade. The largest producers are the mines at Mineville, near Port Henry, New York, from which 9,150,000 tons are estimated to have been obtained.

The separation of iron-ores is practiced to a considerable extent in the Lake Champlain district, the leaner ores obtained in mining being crushed and jigged. In 1888 there was shipped of this dressed ore 54,048 gross tons in addition to what was used locally.

The total product of the Lake Champlain district in 1888 was as follows:

	Gross tons.
From mines near Port Henry, N. Y.,	419,009
From mines of Chateaugay Ore and Iron Co.,	250,500
From mines near Crown Point,	74,910
From other mines, estimated,	45,000
Total,	789,419

The total product for 1887 was 768,852 tons.

This makes the Lake Champlain district outrank the Cornwall ore-banks for the year 1888.

IV. The New Jersey Mines.

New Jersey fell considerably behind former years in 1888, the total being 447,738 gross tons, as against 547,889 gross tons mined in 1887. The aggregate product of the New Jersey mines since their opening, dating back into the last century, is estimated by Prof. Geo. H. Cook at 16,500,000 gross tons.

Most of these ores are magnetites, and a majority of them carry considerable sulphur, while in others the phosphorus is high; but New Jersey also produces Bessemer ores. The unique deposit of Franklinite (a mixture of oxides of iron, manganese and zinc) is generally included among the iron-mines of New Jersey.

The decline of the New Jersey mining industry is traceable to competition at or near the seaboard with foreign ores, which average higher percentages of iron than the New Jersey ores, to the expense of mining and high royalties, and to the necessity of roasting many of them to reduce or eliminate the sulphur.

The New Jersey iron-ore mines have in the past decade produced the following quantities:

Year	Gross tons.
1879,	488,028
1880,	745,000
1881,	737,052
1882,	932,762
1883,	521,416
1884,	393,710
1885,	3 0,000
1886,	500,501
1887,	547,889
1888,	447,738
Total,	5,644,096

V. Missouri.

A district which has become quite prominent as an iron-ore producer is that portion of Missouri in which the Iron Mountain and Pilot Knob are located, but the quantity of ore, chiefly specular, obtained from this district has lately been less than is generally supposed. The total amount shipped in 1888 was 160,000 gross tons, and in the forty or more years during which iron-ore has been taken from this district, a total of about 4,000,000 tons has been obtained, of which 70 per cent. came from the Iron Mountain.

VI. The Southern States.

The rapid increase of iron production in the South has naturally given great importance to that region as an ore-producer, but no definite data have been collected thus far to show its true position by comprehensive statistics. Some of the leading mines supply figures, but for others these are unobtainable. We may, however, deduce approximately the quantity of ore used from the pig-iron output for the year, allowances being made for the shipment of ores from one State to another. A calculation made in this way shows that in 1886 the States of Virginia, North Carolina, Georgia, Alabama, Tennessee and Kentucky produced about 1,750,000 gross tons, while in 1887 the output slightly exceeded 2,000,000 tons. The consumption of ore in these States in 1888 approximated 2,500,000 gross tons. Of this amount fully 70 per cent. was red hematite fossil-ore, about 25 per cent. brown hematite, and probably 5 per cent. magnetite.

Alabama contributed in 1888 about 1,000,000 gross tons, Tennessee about 615,000 tons, and Virginia between 450,000 and 500,000 tons.

VII. Ohio.

Ohio, although a very large consumer of iron-ores, produces but a comparatively small quantity. The total quantity of carbonates and hematites mined in that State during the last three years was as follows:

Year.	Gross tons.
1886,	344,484
1887,	377,465
1888,	253,352

VIII. Minor Sources.

For the remaining sections of the country, which produce more or less iron-ore, a brief reference will suffice.

Maine, Massachusetts, Connecticut and eastern New York furnish considerable amounts of brown hematite, chiefly for local consumption. Magnetite and carbonate are also obtained in eastern New York, and a large mining industry in red hematite and fossil-ores found there is carried on in the northern and central part of that State.

All the various classes of ore are liberally distributed throughout Pennsylvania; and the total product, if it could be obtained, would add to that of Cornwall a large amount of other magnetites, as well as red hematites, red fossil-ores, limonites and carbonates.

Brown hematites and carbonates are mined in Maryland, and some bog-ores are obtained from Delaware and from the eastern shore of Maryland. A small amount of fossil-ore is mined in West Virginia, and some brown hematite in Texas. In addition to the Wisconsin mines embraced in the Lake Superior district, that State yields some fossil-ores and brown hematites, and excellent red hematites are also obtained in central Missouri.

Iron-ores are mined in Colorado, Oregon and Washington Territory for local consumption. The total for Colorado approximates 50,000 tons; the other two regions are small producers.

IX. Foreign Ores.

Imported foreign ores form an important feature of the supply for our iron-works. In 1887 they amounted to 1,194,301 gross tons, or nearly 10 per cent. of all the ore supplied to our blast-furnaces; but in 1888 the importations of iron-ore decreased nearly one-half, and the amount, 587,470 gross tons, was less than 5 per cent. of the

total amount of iron-ore consumed. The importation, 1,039,433 tons in 1886, and also those in 1881 and 1882 respectively, were greater than in 1888.

Authentic records, obtainable only since the year 1879, show for the past ten years an aggregate of 6,340,776 gross tons, distributed as per the accompanying table:

Year.	Gross tons.
1879,	284,141
1880,	493,408
1881,	782,887
1882,	589,655
1883,	490,875
1884,	487,820
1885,	390,786
1886,	1,039,433
1887,	1,194,301
1888,	587,470
Total,	6,340,776

X. Conclusions.

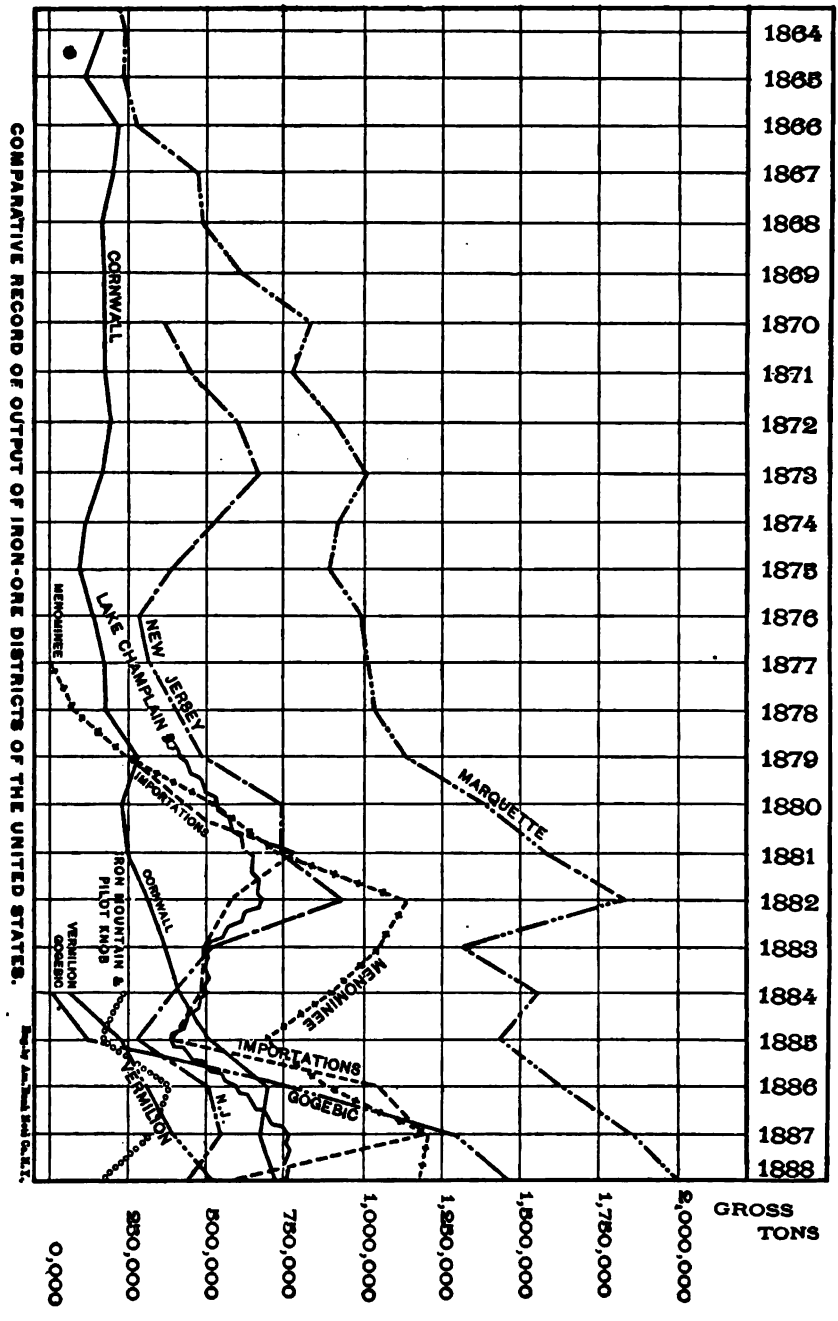
From the foregoing data it is possible to arrange the various sources of iron-ore supply according to their output in 1888. It is not claimed that the relative order or the quantities mined are absolutely correct, but they are as nearly so as it is possible to make them without actual returns from numerous small mines.

Relative Importance of Various Districts in 1888.

Order.	Name.	Gross tons mined in 1888.
1.	Marquette range, Lake Superior region,	1,921,525
2.	Gogebic range, " " "	1,424,762
3.	Menominee range, " " "	1,165,039
4.	Alabama (estimated),	1,000,000
5.	Lake Champlain district, New York,	789,419
6.	Cornwall ore-hills, Pennsylvania,	722,921
7.	Other Pennsylvania mines (estimated),*	700,000
8.	Tennessee (estimated),	615,000
9.	Foreign ores,	587,470
10.	Vermilion range, Lake Superior region,	511,953
11.	Virginia (estimated),	450,000
12.	New Jersey,	447,738
13.	Ohio,	253,352
14.	Missouri,†	160,000

* The total output of Pennsylvania, including the Cornwall ore-hills, would probably make that State rank as third, and next to the Gogebic range.

† Shipments from Iron Mountain and Pilot Knob only.


COMPARATIVE RECORD OF OUTPUT OF IRON-ORE DISTRICTS OF THE UNITED STATES.
1864
1865
1866
1867
1868
1869
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
1888
0,000
250,000
500,000
750,000
1,000,000
1,250,000
1,500,000
1,750,000
2,000,000
GROSS TONS
CORNWALL
MENOMINEE
LAKE CHAMPLAIN &
NEW JERSEY
IMPORTATIONS
MARQUETTE
IRON MOUNTAIN &
PILOT KNOB
VERMILION
GOGEBIC
N.J.

The important position which various districts hold as contributors to our mineral wealth will also appear from the total amount of ore which is estimated to have been taken from them, thus:

Total Estimated Output to Date.

	Gross tons.
Marquette range, Lake Superior region,	27,130,419
New Jersey (estimated),	16,500,000
Lake Champlain district,	15,147,000
Cornwall ore-hills,	9,395,922
Menominee range, Lake Superior region,	8,594,037
Iron Mountain and Pilot Knob, Missouri,	4,000,000
Gogebic range, Lake Superior region,	3,586,876
Vermilion range, Lake Superior region,	1,498,209

The above figures merely indicate the quantities taken from such sources as could be traced. In comparing them time must be taken into consideration as a factor, for iron mining has been followed many years in the Lake Champlain district, in New Jersey, at the Cornwall ore-hills, etc., while the industry of Menominee range is but eleven years old, and that of the Gogebic and Vermilion ranges but five.

The foregoing history is graphically represented on the accompanying chart, showing the quantities of iron-ore mined from year to year, in the Marquette, Menominee, Gogebic and Vermilion ranges of the Lake Superior region; Cornwall, Pa.; New Jersey; Lake Champlain, New York; and the Iron Mountain and Pilot Knob mines, in Missouri; and also the importations of foreign ores. To indicate the degree of statistical accuracy which can be claimed for this summary, a few remarks should be added concerning the data from which it has been compiled. The Marquette range commenced shipments in 1854, but did not reach 200,000 gross tons until 1863. The Cornwall ore hills have been operated for 150 years; but the annual reports of the Cornwall Ore-bank Co. date only from 1864, in which year the output, 166,000 tons, was for eleven months. The authentic records of the New Jersey mines cannot be traced back continuously beyond 1870, although ore was mined from them prior to 1800. A fire which destroyed the books of one of the large producers prevents tracing the output of the Lake Champlain mines further back than 1879. Shipments commenced from the Menominee range in 1877; from the Gogebic and Vermilion ranges in 1884. Importations of foreign ore were not reported before 1879. Records of the product of the Iron Mountain are in existence for a number

of years, but those of the other neighboring mines were destroyed by fire, hence continuous data are obtainable as far back as 1884 only. It has been impossible to collect correct figures from the numerous mines which go to make up the total of other districts.

THE CONCENTRATION OF IRON-ORE.

BY JOHN BIRKINBINE, PHILADELPHIA, PA., AND THOMAS A. EDISON, LLEWELLYN, N. J.

(New York Meeting, February, 1889.)

THE *Transactions* already contain many valuable papers on the subject of ore-concentration, but with reference more to the treatment of other ores than those of iron.

In this country much money, labor and thought have been devoted to the enrichment of iron-ores by roasting to drive off sulphur and carbonic acid, eliminate water, or make the ore more friable, and by washing and screening to remove the clay and sand from earthy ores. In concentrating magnetites and hematites more has been done than is generally believed; but, unfortunately, the results up to the present time have not been encouraging. On the contrary, so many attempts have proved unsatisfactory that any system of concentration is now unjustly looked upon with doubt, and the problem presented to individual mines must therefore be approached with caution.

For many years the magnetites of the Adirondack region of New York State have been roasted, reduced to proper size by means of gang-stamps, and jigged on screens in water. Latterly plunger-jigs and rotary-jigs have been introduced, and in some places crushers and rolls have taken the place of stamps. In one instance in that region a plant for reducing the ore by crushers and rolls and separating it by magnetism was put up to treat a titaniferous ore; but, the titanium occurring as ilmenite, the result was unsatisfactory, and the plant is now being reconstructed at another mine, where apatite and silica are to be separated from the magnetite. A large concentrating works was erected at Negaunee, Michigan, in the Marquette region, Lake Superior, to enrich the leaner specular ores. This consisted of crushers, Cornish rolls and rotary-jigs. The results obtained were not satisfactory, and it has been abandoned.

Another plan to treat the lean magnetites from a large dump-pile at Republic, Michigan, by means of pulverizers and water-jigs also

proved unsuccessful, but the parties who constructed it are reported to have lost little money, owing to the fact that others stood ready to pay them a good bonus for the privilege of hand-picking the dump-pile at a fixed rate per ton of ore won. This dump-pile, which is now being hand-sorted, is estimated to contain 2,000,000 tons of ore and rock. There were separated from this mass in 1888 some 21,000 tons of merchantable ore, which, after paying 30 cents royalty and $1.85 freight, netted the operators a profit of 50 cents a ton. The Bessemer ore as shipped yielded 62 per cent. of iron. About two-thirds of it was specular, and one-third magnetic. The contractors expect to ship 40,000 tons from this pile in 1889, and it is estimated that an equal quantity can be obtained for ten years to come. A similar plan of cobbing the dump-piles is in operation in other Marquette mines also. Nearly 30,000 tons of hand-picked ore have been shipped in the last three years by one company formed for that purpose.

These statements and others which immediately follow are offered to indicate that the importance of re-working dump-piles is becoming appreciated. By reason either of imperfections in the earlier operation, or of the impracticability of careful sorting underground, the percentage of merchantable ore in the dump-piles in many mines represents a very serious loss to the operator.

At the Champion mine, in the Marquette region, hand separation has been carried on for some time, but within the last eighteen months a complete system has been adopted which is giving very satisfactory results.* The ore as raised from the mine is either No. 1, No. 2, dump, rock, or waste. The No. 1 ore is mine-sorted, and must be absolutely clean, usually carrying 66.5 per cent. or more of iron. No. 2 ore varies from 63.5 to 66.5 per cent. of iron. The remainder of the ore is carried to sorting-floors, and dumped into a series of stalls, the rock and waste going directly to the rock- and waste-piles. In each of these stalls two men and a boy have two wrought-iron cars, into which the ore is carefully spread by a shovel, and the various grades are picked out by hand. Some of this ore sorted from the stalls is No. 1 "Champion," some is No. 2 "Chelsea," and two inferior grades are also produced: No. 3, "Suffolk," carrying 60.00 to 63.5 per cent. of iron, and No. 4, "Norfolk," yielding 57.0 to 60.0 per cent. To indicate how close this hand-separation

* For the details of the method pursued in the Marquette region we are indebted to Mr. Richard A. Parker, M.E., of Marquette, Michigan.

may be made, we may cite two cargoes, sold as "Norfolk" grade, to yield 57 per cent. of iron. The actual sampling of the ore after delivery showed for the two cargoes 57.09 and 57.12 respectively.

In six months, July 1st to December 31st, 1888, about 43,000 tons of material were thus hand-sorted, the percentages of the various grades being as follows: No. 1, 17 per cent.; No. 2, 11 per cent.; No. 3, 31 per cent.; No. 4, 16 per cent.; rock, which includes everything below the "Norfolk" grade, 25 per cent. The quantity of material handled per man per day is approximately seven gross tons, from which about five gross tons of ore are selected at an average cost of 38.6 cents per ton for labor. The price paid per ton of selected ore varies according to grade. The average rate of wages paid per day during the time above mentioned was $1.75.

The perfection of the system of hand-sorting is due to the plan of organization, and also to the thorough inspection by skilled assistants, whose compensation is made up by assessing the various contractors in proportion to the amounts of ore they deliver. An experiment in magnetic separation is about being tried at this mine.

At the Iron Mountain, Missouri, wet concentration is now in use, but up to the present time no reduction in the size of the ore is made by crushers or other means, the practice being to hydraulic the dump-piles or wash the earthy material from the ore, size it in screens, and pass it through jigs arranged for each size of material. A detailed description of the method pursued appears in the *Journal of the United States Association of Charcoal Iron Workers*, vol. viii. p. 38.

At the magnetite mines in Putnam county, New York, there are two concentrating plants, one consisting of crushers, rolls and plunger-jigs,* which is now being altered so as to separate the ore by magnetism, and a new plant in which the ore is reduced by Ball stamps, washed, and passed through rotary-jigs. It is understood that a magnetic separator also will be added to this plant.

At present the largest producer of separated ore is the Chateaugay Ore and Iron Company, which at its mines in Clinton county, New York, crushes the leaner material obtained in mining, and treats it by water-jigs, the product being a regular portion of the ore mixture of some of our prominent eastern blast-furnaces making Bessemer pig. That the material produced is satisfactory has been

* Described by Mr. A. F. Wendt, *Trans.*, xiii., 35.

demonstrated by Mr. E. S. Moffat, who has used as high as 50 per cent. of his ore burden of Chateaugay separated ore, and each year has increased the consumption. He kindly supplies the following figures as the amount of separated ore used at the furnaces of the Lackawanna Iron and Coal Company, at Scranton, Pa.

	Gross tons.
In 1886,	4,445
" 1887,	9,765
" 1888,	22,173

Mr. A. L. Inman, general manager of the Chateaugay Ore and Iron Company, supplies the following data:

"Prior to 1886 we made separated ore in a comparatively small way, which was used in the production of blooms, but in 1886 we commenced its production with more vigor, making, in 1886, 23,885 tons; in 1887, 47,324 tons; and in 1888, 54,048 tons."

These figures indicate that a market is being found for this finely comminuted ore. The holes in the screens at the separating plant of the Chateaugay Ore and Iron Company, at Lyon Mountain, New York, are $\frac{5}{16}$ of an inch in diameter, but necessarily a large portion of the material is much finer. A hand-trial of the crushed ore before it passed to the jigs showed the average to be 30 mesh; of the total amount only 14 per cent. by weight remained on a 4-mesh sieve, and less than one-half of the entire material was held in a screen of 16 meshes to the inch, while 8.3 per cent. passed through 100 or finer mesh screens.

It may naturally be expected that as a rule blast-furnace managers will object to the fine material, but it is questionable whether many of the ores now used successfully in the blast-furnace are not as finely divided as separated ores. The softer hematites, which may be shoveled up like mud, undoubtedly are speedily converted into sand by drying out in the shaft of the blast-furnace.

These soft ores are much esteemed by the blast-furnace managers of the Northwest, and where the desire is to drive a furnace, the lump ore is often discarded for this fine material; therefore, independently of the character of the ore, we may question whether its finely divided state should not commend rather than condemn it for blast-furnace use.

In crushing any ore a considerable amount of fines is invariably made which pass into the furnace. As to the effect of the separated ore in the blast-furnace, Mr. Moffat wrote on September 1st, 1888:

"My experience has been chiefly with concentrated Chateaugay (Lake Champlain) ore, and we have had so little trouble with it that I feel almost justified in saying that I do not know of any difficulties in its use up to say 50 per cent. of the ore mixture, which is the most I have tried. For the past two months (June and July) we have been running the four Scranton blast-furnaces, which are in operation, on ½ concentrated Chateaugay ore. They have worked just as well as when we were using ordinary furnace ores, no increase of pressure, no irregularities, and no trouble of any kind. A few days since the quantity of concentrated ore in use on our No. 1 furnace (73 feet high × 20 feet bosh), was doubled, running it up to ⅓ of the ore charge. The only other change made was a slight decrease of limestone; the speed of the blowing-engines and all other conditions being kept the same as before. I rather expected some increase of pressure at the tuyeres, but such has not been the case. The furnace was working well before the change was made, and has worked just as well since. The fuel used is ⅓ coke and ⅔ Lackawanna anthracite coal.

"During 1887 we ran our No. 5 furnace (70 feet × 19 feet) for several weeks on a 'burden,' of which 50 per cent. was concentrated Chateaugay ore. When we made this trial I anticipated a considerable increase in pressure of blast, and in order to meet this increased the proportion of coke to one-third. I afterwards concluded that this increase of coke was unnecessary, as the pressure of the blast went down considerably. The furnace worked well on 50 per cent. concentrated ore and showed no peculiarities.

"None of the concentrated Chateaugay ore is coarser than what will pass through a quarter-inch hole, and most of it very much finer. As different blast-furnaces work differently, I would recommend parties commencing the use of concentrated magnetite to try a small proportion at first, say $\frac{1}{12}$, and then gradually increase."*

Interesting data concerning the operation of the plant at Lyon mountain, where this ore was prepared, analyses, etc., are found in a carefully written and valuable paper by Messrs. Maynard and Kunhardt in the *School of Mines Quarterly*, vol. ix., No. 2; and a full description of the plant was presented in a paper by Mr. Fred. S. Ruttman, "Concentrating Magnetite with the Conkling Jig at Lyon Mountain, New York" (*Trans.*, xvi., 609).

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* *Jour. U. S. Ass'n. Charc. Iron Workers*, vol. viii., pp. 25 and 26.

The crushing system in use is also described in a paper upon the "Blake System of Fine Crushing" (*Trans.*, xvi., 753), by Mr. Theodore A. Blake, who gives the cost of crushing and sizing the ore at 34 cents per ton of crude material.

In all systems of concentration the comminution and sizing of the material to be treated are of primary importance; and although the various methods will not be discussed in this paper, it may be remarked that the degree of fineness to which an ore will be crushed for separation, and the special machinery employed, is affected,

1st. By the size of the grains or crystals in the crude ore.

2d. By the foreign matter which is to be eliminated, and its physical condition.

3d. By the purpose for which the concentrate is to be used.

4th. By the condition of the ore and the method employed for separation.

If the ore to be treated is a magnetite with large crystals, or if the object of separation is to remove siliceous material only, a coarser sizing can be employed than in other cases; for practically, complete elimination of silica is not at present essential, and in some ores a system of mechanical sorting or cobbing, treating pieces from fist to walnut size, may be practicable. If, however, the ore is dense, and the crystallization or granulation fine, or if apatite is to be removed, the reduction of size must be carried further, so as to separate, as far as practicable, each particle of magnetite from the other materials. In dephosphorization by mechanical means a *few hundredths* of one per cent. of phosphorus will determine whether the ore will be within or outside of the Bessemer limit, and hence influence its commercial value.

The extent to which an ore is to be crushed will also be influenced by the demand for certain sizes or by the objection to others.

The condition of the crude ore will materially influence the machinery to be employed, a dry or a thoroughly wet ore being easier to manage than one which is damp or partially saturated with moisture.

An experiment is about to be made in crushing ore, heated to such an extent as is believed will cause the minerals composing it to weaken on their cleavage planes, thus facilitating the granulation.

The problem, therefore, is to be determined specially for each ore to be treated, and consequently the particular form of crusher, whether jaw, rotative or centrifugal, or the combination of crushers,

stamps or rolls, and the system of sizing must be such as to meet the particular case under consideration.*

In a test made at the Edison Laboratory of Samson (Humboldt, Mich.) magnetic ore which had passed through shaking screens of 20 meshes to the inch, the quantities stated below were refused by sieves of the various meshes given. This ore was reduced by the Gates crusher and Cornish rolls.

Meshes to one inch.	Grammes refused.	Percentage of total.
30	6.060	2.32
40	17.100	6.55
50	8.900	3.44
60	21.500	8.24
70	17.350	6.65
80	12.350	4.73
90	13.500	5.18
100	22.350	8.57
110	12.350	4.73
120	5.700	2.19
150	12.350	4.73
160	24.650	9.45
200	20.500	7.86
finer than 200	66.150	25.36
Total,† . . .	260.810	100.00

A test made by Mr. W. H. Hoffman at Bayonne, New Jersey, on Croton magnetic ore reduced by a Sturtevant mill, gave the following proportions of sizing:

Meshes to one inch.	Pounds refused.	Per cent. of Total.
2 to 6	5¼	30.43
6	1½	8.69
10	1	5.80
16	1¼	7.25
20	1¼	7.25
50	2¾	15.94
No. 60 and smaller,	4¼	24.64
Total,‡ . .	17¼	100.00

* In the monograph on the "Dressing of Non-Bessemer Ores," by Messrs. Maynard and Kunhardt, to which reference has already been made, a series of crushing-tests is described, and the results of sizing raw and calcined ore are given in tables which offer the most complete information on the subject at present obtainable.

† Apparent average, 125 meshes.

‡ Notwithstanding the coarse sizing this material approximated an average of 30 meshes per inch.

Magnetic Concentration.

At present, much attention is devoted to the concentration of iron-ores by means of magnetism, a method which, in view of the dates and numbers of patents already issued, cannot lay claim to novelty. The patents issued by the United States Patent Office appear to cover nearly every form of apparatus, but new designs are being rapidly added.

The earlier inventions were adapted for separating iron from brass and other filings in machine-works, etc., and also for the removal of wire from grain. Attempts to separate some of the magnetites of New York and New Jersey were also made upwards of twenty years ago, and experimental work was done in separating the magnetic iron-sands of the St. Lawrence river and Long Island Sound.

The various inventions for separating iron-ores by magnetism may be grouped under two general heads, viz.: those which have permanent magnets and those which have a fixed or an alternating current of electricity passed through magnets from a dynamo. These groups may again be classified into such as receive the ore on tables or belts which pass under or over, or which are traversed by magnets; those which consist of magnetic rolls or drums either receiving the ore on their surface or revolving over the ore; and those which by magnetic influence draw the magnetic portion from a falling mass, altering its trajectory so as to separate it from the gangue material. They may be further subdivided into those which treat the ore dry and those which immerse the material for concentration in water.

While in some ores a concentration answering most economical purposes may be readily made, the physical structure of the ore prevents a more perfect separation. In one New York magnetite, a microscopic examination, which Mr. E. K. Landis made, of the ore after it passed the crushers, rolls and screens, proved that, however fine the grain, it in many cases consisted of pieces about $\frac{1}{100}$ of an inch in diameter, showing, under the microscope, magnetite, magnetic pyrites, quartz, and sometimes hornblende.

Dr. Eugene Hussak, in his monograph on "The Determination of Rock-Forming Minerals," says: "If several ferriferous mineral species occur in the rock to be examined, *e.g.*, magnetite, ilmenite, augite, biotite, olivine, etc., they can be separated from each other by varying the strength of the current passing through the electromagnet. At first, two elements are used, then four, six, eight, and finally ten. Doelter has shown that the minerals can be arranged

in a series according to their different powers of being attracted." He gives thirty-three minerals and indicates their order of attractability by magnets.

Dana states that magnetite is strongly magnetic and sometimes possesses polarity, and that from the normal proportions of one part of FeO to one Fe_2O_3 there is occasionally a wide variation, and thus a gradual passage to the sesquioxide Fe_2O_3. He instances analyses from Bengal, India, in which the proportion of Fe_2O_3 to FeO was nearly three to one, in an ore which was strongly magnetic and columnar, while another ore from the same locality, in which the proportion of Fe_2O_3 to FeO was three and a quarter to one, was granular and not polar magnetic. He also says that hematite is sometimes attracted by a magnet and, occasionally, even magnetipolar. Martite, which is supposed to be a pseudomorph, mostly after magnetite, is reported as non-magnetic or only feebly so, and menaccanite as slightly influencing the magnetic needle.

The presence, therefore, of certain of these minerals will materially affect the results obtained, as we naturally look for a better separation when the material in the gangue is but slightly attracted by the magnet or practically insensible to its influence.

Mr. Ellis Clark, Jr. (*Trans.*, ix., 451), describes a magnetic separator in use at Pribram, Bohemia, in which zinc-blende is separated from spathic iron-ore, previously roasted to make it magnetic, and reduced to grains less than one mm. (probably 25-mesh).

The writer knows of at least ten forms of magnetic ore-separators with which experiments are now being made, and five of these may be considered as having been operated on a commercial scale.

In view of the experimental state of some of the apparatus and the fact that the patents for certain features are now pending, it is impossible at this time to attempt a description of any, except such as have been operated on a scale approximating commercial utilization. Hence, but four general forms will be considered.

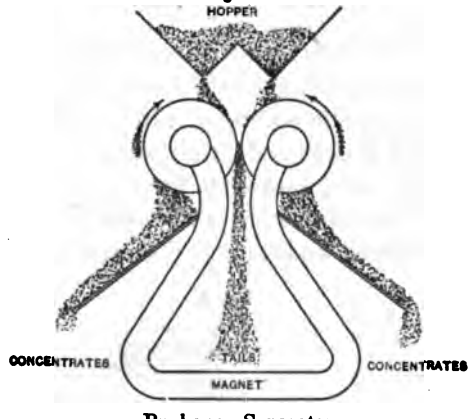
About eight years ago attention was drawn to magnetic separation by the treatment of some of the sea-sands upon our coasts, by passing the material between rolls, the adjacent portions of which were in a magnetic field, while the opposite parts were without this field. The iron rolls revolved on journals carried on insulated standards wound with copper wire. By connecting these wires with a battery (latterly with dynamos) the standards became electro-magnetic of opposite polarity and the rolls were charged thereby, making magnetic field between the rolls. The ore being fed on the rolls which revolved toward each other, was carried around the magnetic

portion, adhering until it passed beyond the magnetic influence, while the non-magnetic particles dropped between the rolls.

The Buchanan separator, which consisted of such a pair of rolls and a large horse-shoe magnet properly wound, as shown in outline, Fig. 1, was employed in separating magnetite from the fine sea-sand from the shores of Long Island Sound, and an extensive plant was sent to New Zealand, where the sea-sand carries a remarkable amount of finely comminuted magnetite.

A pair of these rolls has lately been operated at the Croton magnetite mines, near Brewster, N. Y., by the Messrs. Cheever, to prepare concentrates from the waste-piles of lean ore. The ore, a

Fig. 1.



Buchanan Separator.

dense magnetite, is reduced by jaw-crushers and Cornish rolls so as to pass through 16-mesh screens. Average analyses, made by E. K. Landis, of the lean ore left on the waste-piles, show the general composition of the crude ore, and the concentrates and tailings obtained from this ore as follows:

Concentrations of Croton (Theal) Ore, N. Y., by Buchanan Magnetic Rolls.

	Crude ore.	Concentrates.	Tailings.
Iron,	37.968	64.554	13.207
Silica,	29.30	5.350	50.20
Phosphorus,	0.383	0.050	0.492
Sulphur,	0.522	1.253	0.924
Iron,	31.28	62.56	4.66
Silica,	35.50	6.825	58.05
Phosphorus,	0.271	0.058	0.527
Sulphur,	0.784	1.320	0.945

The increased amount of sulphur in the concentrates, is probably due to the pyrite being magnetic.

Some years ago, the reducing-plant was operated to prepare ore for plunger-jigs, and the results obtained were as follows:

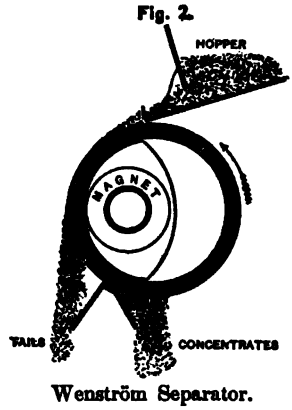
Concentration of Croton (Theal) ore by Plunger Jigs.

	Fine Jigs.	Coarse Jigs.
Crude Ore:		
Metallic Iron	36.48	36.48
Concentrates:		
Metallic Iron	65.56	58.78
Phosphorus	0.047	0.129
Sulphur	0.97	0.83
Tailings:		
Metallic Iron	14.31	22.16

Another machine consists of a single roll with alternate strips of magnetic and non-magnetic material forming its periphery.

The Wenström magnetic separator, which is of Swedish origin,* has a stationary field magnet and an armature barrel consisting of a number of soft iron bars, separated from one another by a non-magnetic material—in this case strips of wood. The whole is bound together by non-magnetic end-rings. The bars are cut away alter-

Fig. 2.



Wenström Separator.

nately on the inside to make one bar project only towards the north poles of the magnet and the next only to the south poles. This gives each succeeding bar opposite magnetism. On each of the four sections of the magnet are wound fifteen pounds of copper wire. An Edison dynamo furnishes a current of ten amperes and thirty-three volts. The ore is fed to the barrel by means of a hopper, as

* A description of this machine will be found in Mr. R. A. Cook's paper on "The Wenström Magnetic Separator," at page 599 of the present volume.

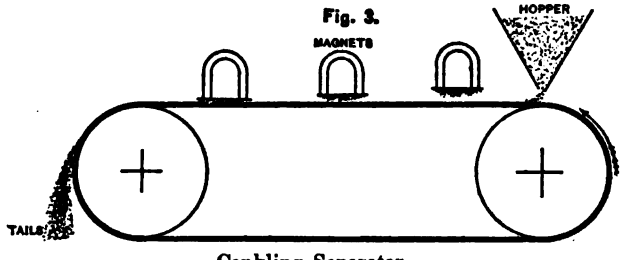
shown in outline, Fig. 2, the cylinder turning in the direction of the arrow. The magnetite adheres to the bars of the barrel and is carried downward past the first delivery shute. Below the machine the bars, departing from the influence of the electro-magnet, which is placed eccentrically, lose their power to hold the particles of magnetic iron-ore and they drop off. The particles of rock in the ore being non-magnetic, drop from the barrel almost immediately and fall on the first shute shown in the engraving. Experimental tests on the Wenström machine with various ores gave results as follows:

Concentrations by the Wenstrom Magnetic Separator.

	Crude ore.	Concentrates.	Tailings.
Beach Glenn, New Jersey,			
Iron, ...	53.73	61.53	3.25
Phosphorus, ...	0.023	0.006	
Chateaugay Ore & Iron Co., N. Y.,			
Iron, ...	40.99	59.80	1.62
Port Henry, New York, New Bed, Witherbees, Sherman & Co.,			
Iron, ...	41.6	64.0	15.10
Port Henry, New York, Old Bed, Witherbees, Sherman & Co.,			
Iron, ...	59.1	61.8	14.3
Phosphorus, ...	1.249	0.847	4.825

The Conkling magnetic separator is a belt-machine of the general form indicated by Fig. 3, which merely shows the principle and

Fig. 3.



Conkling Separator.

not the detail. The ore is fed on a belt and carried along under a series of belts, running at right angles to the first. These cross-belts pass between the magnets and the ore lying on the distributing-belt, and may be placed at varying distances from the latter. As the ore, reduced to the proper size, passes along on the distributing-belt, the magnetic-belts, which may be influenced by magnets of different powers, pick up and carry to one side the magnetic particles of the ore, while the non-magnetic portion of the gangue is carried

off as tailings. The results obtained by treating some ores by this machine are reported as follows:

Concentrations by the Conkling Magnetic Separator.

Test.	Crude Ore.		Concentrates.			Tailings.	
	Iron. Per cent.	Phosphorus Per cent.	Iron. Per cent.	Phosphorus Per cent.	Silica. Per cent.	Iron. Per cent.	Phosphorus Per cent.
I.	53.80	0.010	70.80	0.006		21.00	0.10
II.	62.04	1.146	70.29	0.153		not det.	not det.
III.	58.00	0.100	79.47	0.022	1.27	"	"
IV.	39.00	0.038	66.24	0.005		"	"

I. Port Henry, N. Y., New Bed, Witherbees, Sherman & Co.
II. Platt Mine, N. J.
III. Mt. Hope Mine, N. J., Thomas Iron Co.
IV. Lehigh Mountain, Pa.

Another machine which has been operated experimentally on a commercial scale is called the "Monarch." In this the ore is fed on one belt, from which the magnetic particles are picked by a second belt placed above the first, but carrying the ore in the same direction; this belt is equipped with magnets of alternate polarity, to obtain, if possible, more perfect separation of the material as it passes the varying poles.

Analyses of results reported from this machine are as follows:

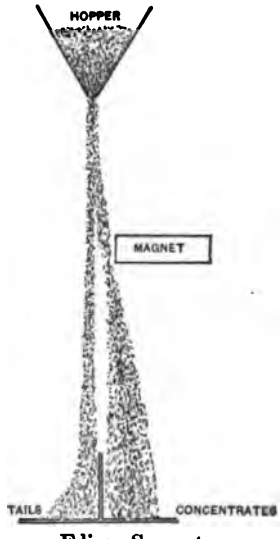
Concentrations by the Monarch Magnetic Separator.

Test.	Crude Ore.		Concentrates.		Tailings.	
	Iron. Per cent.	Ph'sph'rus Per cent.	Iron. Per cent.	Ph'sph'rus Per cent.	Iron. Per cent.	Phosphorus. Per cent.
I.	36.23		66.98		8.12	
II.	35.48	0.158	61.40	0.042	5.69	1.45
III.	64.29	0.723	69.80	0.220	5.05	5.30
IV.	54.30	0.089	63.70	0.044	12.80	0.408
V.	58.00	2.256	68.50	0.407	9.20	10.787

I. Chateaugay Ore and Iron Co., N. Y.
II. Little River, N. Y.
III. Forest of Dean, N. Y.
IV. Witherbees, Sherman & Co., N. Y., New Bed.
V. Witherbees, Sherman & Co., N. Y., Old Bed.

The Edison unipolar non-contact electric separator differs from the forms described in that it has no moving parts. Except such facilities for altering the relative position of the parts as are essential for adjustment in treating different ores, or are required to secure certain results, all parts of the apparatus are fixed. The separator, which is illustrated by Fig. 4, consists simply of a hopper, a magnet and a partition to separate the concentrates and tailings into different receptacles. The illustration shows but one hopper, but in practice the ore can pass on each side of the magnet, thus doubling the capacity. The simplicity of the construction, which is the

Fig. 4.



Edison Separator.

result of patient and thorough investigation of many different designs and methods, will commend itself.

The ore after being properly crushed and sized is placed in hoppers, from which its discharge is controlled by bars closing slots which extend the length of the hopper. These slots are made adjustable so as to suit the size to which the ore has been reduced. The hoppers are adjusted to appropriate heights above the magnet. The magnet in the apparatus exhibited at the Edison Laboratory, Llewellyn, N. J., is a mass of soft iron 6 feet long by 30 inches wide and 10 inches thick, weighing 3400 pounds, and wound with 450 pounds of copper wire, the coil being connected with a dynamo

consuming 2½ horse-power and requiring a current of electricity of 16 amperes and an electromotive force of 116.5 volts.

The material falling from the hopper passes the face of the magnet, but does not touch it. The distance of the magnet from the vertical plane of the falling material is so chosen that its attraction causes the magnetic to separate from the non-magnetic particles sufficiently to alter their direction. By reason of the force of gravity, this deflection of the trajectory, while sufficient to draw the magnetic particles away from the non-magnetic, does not draw them against the magnet, but should any ore accumulate on the magnet it can be instantly dropped by breaking the current. The exact distance, however, is maintained so that none can stick to the magnet. Owing to the altered trajectory the magnetic ore falls upon one side of the partition, which is so adjusted as to secure the best result while the gangue material drops upon the opposite side.

In many ores there are particles of magnetite attached to some non-magnetic material which prevent them being carried over with the concentrates, but cause them to be drawn sufficiently from the vertical to separate them from the tailings, or when the stream of material permits several layers to pass the magnet simultaneously, particles of non-magnetic material may retard the movements of magnetic portions so that they do not pass into the concentrates. In such cases an intermediate grade is collected called the "mugwump," because it is neither concentrates nor tailings. This mugwump may be returned to the hoppers or passed before a second magnet. A series of magnets may be arranged so that the concentrates, mugwumps or tails are each subjected, as in other machines herein described, to repeated magnetic influence, thus insuring more perfect separation, and maintaining the capacity which is a strong point of the Edison apparatus. The hopper in the apparatus exhibited to the Institute at Llèwellyn is 6 feet wide, and the quantity of ore which can be treated per day will depend upon the degree of fineness to which the material is crushed, and the width to which the slot in the hopper is opened. Arranging the slot to pass readily the lean ore from Witherbees, Sherman & Co.'s New Bed at Port Henry, N. Y., when crushed to pass a 10-mesh screen, each side of the magnet will separate conveniently about 150 tons of material daily, making the capacity of the two-face machine 300 tons per day.

The most systematical experimenting upon any one ore has been made by Messrs. Witherbees, Sherman & Co., at Port Henry, N. Y., using various machines, and by Messrs. Cheever with the Buch-

anan magnetic rolls, treating the ores from the Croton mines, but a greater variety of ores have been concentrated by the Edison unipolar machine, among which may be mentioned, Witherbees, Sherman & Co.'s New Bed ore (to separate the magnetite from the siliceous gangue) and Old Bed ore at Port Henry, N. Y. (to reduce the amount of phosphorus by separating the apatite from the magnetite); the Chateaugay Ore & Iron Co.'s ore, at Lyon Mountain, N. Y. (to separate the magnetite from siliceous gangue); Messrs. Cheever's Croton mines, Brewster, N. Y. (to separate the magnetite from the gangue and reduce the phosphorus and sulphur); the Cornwall Ore Bank Company's ore, Lebanon, Pa. (to separate magnetite from silica and reduce sulphur); Mont Alto, Pa. (to separate small crystals of magnetite from a lean ferruginous sand-rock); and the Phœnix Iron Co.'s ore, Jones mine, Berks Co., Pa. (to separate the gangue from the magnetite and reduce the sulphur). Various kinds of waste material from iron and steel works have also been treated to extract the metal. With a modification, the apparatus has been used in treating gold-ore.

The concentrating plant lately erected at Humboldt, Michigan, consists of a large Gates rock-breaker, into which the lean magnetite from the dump-pile, yielding 40 to 45 per cent. of iron, is fed. The material from the crusher passes over a shaking screen into a pair of 16 x 30 Cornish rolls, and from these the crushed ore is sized in screens to 20 mesh, the dust blown out of it, and the product from these screens passes an Edison magnetic separator, where the ore is concentrated. The result is a concentrate carrying from 62 to 68 per cent. of iron, and within the Bessemer limit as to phosphorus. As the plant has but lately been put in operation, the details of its workings cannot be given. The mugwump made is returned by elevators to the feeding bin.

The following are some of the results obtained in treating iron-ores by the Edison separator:

Concentration of New Bed Lean Ore, Port Henry, N. Y., by the Edison Machine.

		Crude ore.	Concentrates.	Tailings.
A. Crushed to 20 mesh,	Iron,	53.2	69.9	7.67
	Phosphorus,	0.03	0.01	0.03
B. Crushed to 10 mesh,	Iron,	50.0	70.0	7.80
	Phosphorus,	0.025	0.018	0.041
C. Above 10 mesh,	Iron,	52.2	66.8	18.70
	Phosphorus,	0.032	0.013	0.085

Concentration of Old Bed Ore, Port Henry, N. Y., by the Edison Machine.

	Crude ore.	Concentrates.	Tailings.
Iron, ...	59.5	69.15	7.10
Phosphorus, ...	1.77	0.41	11.06
Iron, ...	62.0	70.90	9.25
Phosphorus, ...	1.46	0.18	10.54
Iron, ...	64.20	71.20	9.00
Phosphorus, ...	1.39	0.31	11.57

For the purpose of testing the apparatus, ore was selected with a maximum of apatite, and the phosphorus therefore shows largely in excess of the average contents in Port Henry Old Bed ore as mined.

It will be observed that while a considerable portion of the phosphorus has been eliminated it is still above the Bessemer limit. When it is considered how quickly a few stray crystals of apatite will affect the result, or how easily particles of apatite may be embraced as it were by particles of magnetite as they pass the magnet, the delicacy of the operation of removing the phosphorus will be appreciated. Experiments now in progress indicate that a more thorough dephosphorization than given above will be obtained.

Concentration of Croton Ore, N. Y., by the Edison Machine.

	Crude ore.	Concentrates.	Tailings.
Iron, ...	37.97	64.72	11.04
Phosphorus, ...	0.38	0.10	0.97

Concentration of "Jones Ore" of Phœnix Iron Co., by Edison Machine.

	Crude ore.	Concentrates.	Tailings.
Iron, ...	39.507	59.688	32.553
Silica, ...	12.643	5.286	22.143
Phosphorus, ...	0.106	0.010	undet.
Sulphur, ...	0.395	0.157	0.877

This analysis is by Voudy, those of Port Henry ores are by Woodbridge, and those of Croton ores by Landis.

A REVIEW OF THE IRON-MINING INDUSTRY OF NEW YORK FOR THE PAST DECADE.

BY JOHN C. SMOCK, ALBANY, N. Y.

(New York Meeting, February, 1889.)

THE ten years, 1879 to 1888, inclusive, have been notable in the history of iron-making for the great rise during the latter part of 1879 and the earlier half of 1880; for the maximum of production in 1882; for the gradual decline and the reduced product through 1883, 1884 and nearly to the end of 1885; then, for the recovery through 1886 and 1887; and the extraordinary outputs of our iron furnaces during the last three years.

The price of pig-iron might be taken as an index to the condition of the iron-mining industry, and if represented graphically, they would show approximately parallel curves. The mining industry cannot always respond quickly to a rise in price and an increased demand for ore; nor can its production drop simultaneously with sudden falls in price of iron and of ore. Hence, the irregularities of the one are not reflected in the other.

The statistics of the iron mines collected by the United States census, for the year ending June 30, 1880, exhibit admirably their condition at that time. The reports received from mine owners and managers, of the production for the year 1888, furnish the basis for a comparison and supply data for some notes on the changes in the iron-mining industry in New York which have taken place in this period of time.

The iron-ores of New York may be grouped in the following districts:

I. The Highlands of the Hudson (magnetites).
II. The Lake Champlain and Adirondack Region (magnetites).
III. St. Lawrence and Jefferson Counties (red hematites).
IV. Clinton and Wayne Counties (fossil-ores of the Clinton Group).
V. Dutchess and Columbia Counties (limonites).
VI. The Hudson River (carbonates).
VII. Staten Island (limonites).

The ores of the first and second districts, as enumerated, are almost exclusively magnetites; those of the third and fourth are hematites;

those of the sixth are carbonates, as are also some of the ores associated with the limonites of the fifth group.

I. The Highlands of the Hudson.

About forty productive mines have been opened in this mountainous belt of territory between New Jersey, at the southwest, and Connecticut, on the east. Twenty-six of them were in operation during the whole or a part of the census year, the aggregate output for that year amounting to 184,859 tons.* During the decade the historic Greenwood mines have been abandoned; the mines near Warwick also have ceased to be productive; the Todd and the Croft mines, north of Peekskill, the Theal and McCollum and the Brewsters mines, all in Putnam county, have been idle for several years. The Sterling Iron and Railway Company, the Forest of Dean Iron-Ore Company, the Mahopac Iron-Ore Company and the Tilly Foster Iron Mines are the only companies now at work; and they are operating eight mines only, or less than one-third of the whole number producing during the census year. Work has stopped at the smaller mines, and only two away from rail are in operation. Work at the Croton magnetic mines was resumed last autumn to supply concentrates for an experimental direct-process plant.

At the Tilly Foster mine the production was, until lately, subordinated to the great undertaking of removing the cap-rock and pillars preparatory to open pit work. Practically, the lower rates and the less active demand for non-Bessemer ores have resulted in the closing of all those mines whose cost of production is high, either because the vein is small or irregular, or because they are remote from rail transportation.

The production of the Highlands belt in 1888 amounted to 115,000 tons, or only 63 per cent. of the output during the census year.

II. The Lake Champlain and Adirondack Region.

According to the tenth census, the iron-ore mined in 1879–80, in Washington, Essex and Clinton counties amounted to 742,865 tons. At that time ten idle mines were reported.† The Crown Point group and the Port Henry mines were then in active operation, Chateaugay had begun to produce largely, and in Clinton

* Including 672 tons of limonite mined at the Townsend mine, Canterbury.

† The number of mines in this district is variable, according to what definition of a mine is accepted.

county there were several small mines the ore of which was used by the forges then running.

Chateaugay has had an era of growing productiveness and has yielded an aggregate of 1,500,000 tons of ore. The Port Henry mines have fluctuated in their output, falling from 395,000 tons in the census year to 235,000 tons in 1885 and then rising to 428,000 tons in 1887 and 412,000 tons in 1888. Their large deposits of rich ore enable the owners to work them at a low cost and to compete with mines nearer the markets and more advantageously situated. The mines having smaller ore-bodies or distant from railway lines in this district cannot compete and are all closed. The Chateaugay, the Palmer Hill and the Arnold Hill mines are in operation in Clinton county. The Palmer Hill ores raised from the mines of the J. & J. Rogers Iron Company supply the forges of that company. The concentrated ores at Chateaugay go to the forges of the Chateaugay Ore and Iron Company, their bloomary fires being the only ones active in Clinton county. Their future, and that of the mines supplying them, is altogether unpromising.

On the western side of the Adirondack region, the openings at Jayville have been made productive, and the railway has been constructed during the year to Little River, where a large deposit of lean ore has been exploited. There have been no further developments in iron-ore beds elsewhere either in St. Lawrence or in Franklin counties. The mines in Washington county have been idle nearly the whole decade.

The aggregate product of this district, according to reports received from the mining companies, was, in 1888, 812,000 tons, or greater by 70,000 tons than that of the census year.

III. St. Lawrence and Jefferson Counties.

These ores occur in a narrow belt bordering on the west the Laurentian formation of the Adirondacks. Geologically, they might be grouped as Huronian. The Shirtleff and the Tate and Polley, the latter famous half a century ago, have been abandoned during the decade under review. The Keene was closed in 1887. Two new mines have been opened, viz.: the Pike and the Clark. The rich and easily reducible ores of this range are liked by iron-masters, and in spite of the cost of transportation, find a market in the iron-making districts of eastern Pennsylvania and New York. They are often called the Antwerp ores, from the mines near Antwerp, in Jefferson county.

In 1888 six mines produced 110,000 tons, as compared with 94,765 tons in the census year, equivalent to an increase of 16 per cent.

IV. Clinton and Wayne Counties.

The hematites of this geological horizon (the Clinton group of the Silurian) are mined in the vicinity of Clinton, Oneida county, and Sterling and Ontario in Wayne county. Ten years ago there was a long line of openings across Oneida county, comprising twenty separate mines, from which the aggregate amount of ore taken was 85,442 tons. During the decade under review, they have all ceased to be productive, excepting the mines near Clinton, which supply the Franklin Iron Works and the Kirkland furnace. Two mines only in Ontario are in operation and one in Sterling. The Wayne county mines send their output to Charlotte, Elmira and Syracuse. Although the ore is rich and the open-working is in places done at comparatively low cost, the market for the ore is local. The large amount of ore mined at Clinton for the Franklin Iron Works helps to keep the total output of the Clinton ore very nearly as large as it was in the year of the census. The total reported and estimated product in 1888 was 75,000 tons.

V. Dutchess and Columbia Counties.

The brown hematites in this part of the State occur in two well-defined ranges: one, the Fishkill and Clove belt, and the other being that of the valley through which the N. Y. & Harlem R.R. runs. In the two ranges are twenty-five separate mines, of which eighteen were in operation in the last census year. On account of the increasing cost of working, one after another has been abandoned, until at the end of last year, only the Fishkill, Beekman, Clove Spring and Weed were in operation. The total product of the district has fallen from 144,878 tons in the census year to 43,000 tons in 1888. The decrease in output is greater than in any other part of the State, and the outlook for the future is not bright.

VI. The Hudson River.

The history of the Burden carbonate-mines may be said to belong to the decade under review. It was not until 1883 that the Hudson River Ore and Iron Company began to put ore on the market. The production reached 90,000 tons the next year (1884); then declined to 55,000 tons in 1885; and again reached a maximum in

1887, when the output was 142,000 tons. The total product up to 1889 was 350,000 tons. Although not as rich as the ores of the other districts of the State, the geographical position, practically on tide water, and the rail-connections with the furnaces of the Hudson river and of eastern Pennsylvania give to this carbonate-ore advantages, compensating for its lower yield in iron.

VII. Staten Island.

No mines on Staten Island are in operation. The irregularity of the limonite deposits, the leanness of the ore generally, and the necessary cartage to boats or railways, are against steady working, and their final abandonment is threatened. The census reported their total product at 9318 tons.

Summary.

The total product of the iron-mines of the State in 1888 was 1,207,000 tons.* According to the Ninth Census, New York produced 14 per cent. of the iron-ore mined in the country; ten years later, the State produced 1,262,127 tons, or 15.4 per cent., and ranked third in the list of states. In 1886 the product of all the iron-mines in the country, as estimated by James M. Swank, General Manager of the American Iron and Steel Association, was 10,000,000 tons.† In 1887, according to the same authority, it amounted to 11,300,000 tons. New York mines produced in the former year about 900,000 tons, and nearly 1,100,000 in the latter year, or 10 per cent. of the whole. In 1888 the same average proportion was maintained, but the rank changed to fourth, falling behind Michigan, Pennsylvania and Wisconsin.‡

The fluctuation in the totals for the State during the decade have not been so great as might be inferred from the sharp fluctuations in the prices for pig-iron; and the steadiness in the figures for 1886, 1887 and 1888 is remarkable proof of the enduring capacity of the mines of the State. The variation from year to year is not as great as it is in the magnetic iron-ore districts of New Jersey.§

* This sum includes all the returns received from the mining companies and carefully made estimates for three mines unreported.

† *Ann. Statistical Rep. of the Am. Iron and Steel Association*, p. 88.

‡ According to the last report of the American Iron and Steel Association, the total for the United States in 1888 was 12,050,000 gross tons.

§ The product of the iron mines in New Jersey in 1880 was 745,000 tons; in 1885 it had fallen to 330,000 tons, and in 1887 had risen to 547,000. (See *Ann. Reps.* of Prof. George H. Cook, State Geologist, for 1886 and 1887.)

Another notable fact brought out in this comparison is the diminished number of mines producing these totals. At the commencement of the decade, there were about 100 mines at work; last year, only 50 were producing ore.

A RAPID METHOD FOR THE DETERMINATION OF PHOSPHORUS IN CERTAIN ORES.

BY T. REED WOODBRIDGE, PORT HENRY, N. Y.

(New York Meeting, February, 1889.)

THE object of this paper is to bring to notice a method in use at the Cedar Point laboratory of Witherbees, Sherman & Co., for determining the phosphorus in the ore mined at Mineville, N. Y. This ore is magnetite, with a gangue containing crystals of apatite.

In 1886, Mr. H. F. Dawes, then head chemist at the works, suggested that as all the phosphorus seemed to be in the apatite, it could be obtained in solution by dilute nitric acid. After some experimenting the following method was adopted, and has proved invaluable for rapidity and ease; half a dozen determinations requiring very little more time or trouble than one.

From $\frac{1}{2}$ to 10 grammes of ore are weighed into a 150 c.c. beaker; 50 c.c. hot water and 10 c.c. strong nitric acid are added; the beaker is covered and the contents are boiled for $\frac{1}{2}$ hour (10 minutes sometimes proves sufficient; but $\frac{1}{2}$ hour is safer). The bumping which ensues if more than $\frac{1}{2}$ gramme is used need cause no trouble. The hot solution is filtered into a 250 c.c. beaker. The filtration and washing (with hot water) proceed rapidly, as the magnetite itself is but slightly acted upon. After the addition of 15 c.c. ammonia and 15 c.c. nitric acid, the solution is cooled to 45° C., and from 15 to 40 c.c. of molybdic acid solution is added and stirred vigorously. In 5 minutes the precipitate will be down, and the determination can be finished by any good method. At Cedar Point the method more commonly used is weighing as yellow salt. Good washed filter-paper is used, No. 0 Swedish being preferred; and a dozen 3½-inch filters are numbered and dried and weighed at 110° C. These are always kept in stock. The yellow salt is filtered on one of these, washed with dilute nitric acid (2 per cent. by volume); dried for 1 hour at 110° C., and then weighed, 1.63 per cent. of weight being calculated as phosphorus. The filters vary in the two weighings from

0 to 1 milligramme, causing an extreme error of .00163 per cent. in a 1-gramme solution. By using counterpoised filters or a Gooch crucible this error may be avoided. A determination by this method can be made in less than 2 hours; by Emmerton's method in 1½ hours. On fusing the residue from the treatment with dilute nitric acid no phosphorus is obtained, showing that it was all extracted by the dilute acid.

The method used for comparison in the appended tables is as follows:

From ½ to 10 grammes of ore are dissolved in HCl; the solution is evaporated to dryness and heated for 1 hour; redissolved in HCl; evaporated to a sirupy consistency; 20 c.c. to 50 c.c. strong nitric acid is added and the solution is boiled down to 10 c.c. to 15 c.c.; then 40 c.c. of cold water is added, and the solution is filtered and precipitated as above. In Table I. the phosphorus from this solution was determined as yellow salt. In Table II. the yellow salt was dissolved in ammonia, and the phosphorus determined by the magnesia method, precautions being taken for the separation of silica, which rarely proved to have been necessary. The results marked *a* in Table II. were determined by the magnesia method from the yellow salt obtained from the dilute nitric acid solution.

TABLE I.

No.	Dilute HNO_3.	Yellow Salt. HCl.	No.	Dilute HNO_3.	Yellow Salt HCl.
1	.015	.015	11	.672	.686
2	.021	.020	12	.900	.886
3	.030	.032	13	1.087	1.088
4	.088	.087	14	1.143	1.127
5	.100	.095	15	1.645 1.635	1.645
6	.258	.261	16	1.930	1.932
7	.298	.300	17	3.212	3.230
8	.300	.304 .299	18	10.520	10.570
9	.462	.463	19	10.612	10.526
10	.599 .614	.605	20	11.977	11.977

TABLE II.

No.	Dilute HNO_3.	Magnesia HCl.
21	.100 *a*	.097
22	.106 *a*	.109
23	.453 *a*	.466
24	.496	.508 *a*
25	.519	.531
26	.731	.735
27	.770	.779 *a*
28	1.23 1.22	1.23 *a*

a Determined by the magnesia method from the yellow salt.

The results here given were taken at random from a large number

of comparative analyses made during a year or more, the object being to give a few results of both high- and low-phosphorus percentages. Nos. 18, 19 and 20 were taken from the gangue resulting from a magnetic separation of the ore. Silica was looked for in the yellow precipitate; but a trace only was found.

As to the accuracy of calculating the phosphorus from the weight of the yellow salt, there seems to be little doubt, if the amounts of ammonium nitrate and free nitric acid and the temperature of the solution are kept within certain limits. I have repeatedly compared the results of this method with those of the magnesia method and find no practical variation. In experienced hands the yellow salt gives much more uniform results.

I have used the dilute nitric acid on several magnetites and hematites from different parts of the country with some very gratifying results, and hope to present these, both positive and negative, at another meeting.

Previous to the introduction of this method by Mr. Dawes I had not known of its use in connection with iron ore, although it is commonly applied to phosphates and fertilizers; and this paper is presented with the hope that some chemist who has appetizing ore to deal with may be helped thereby.

Discussion.

David H. Browne, Ann Arbor, Mich. (Communication to the Secretary): In regard to Mr. Woodbridge's interesting paper, I would say that I have had experience with this method since 1886. In that year my attention was called by Mr. Brewster, chemist of the Chapin mine, at Iron Mountain, Mich., to a short paragraph in the *Engineering and Mining Journal*, by Mr. Jennings, chemist at the Commonwealth mine, in which the solubility of phosphorus in iron-ores in nitric acid was pointed out.

I have made frequent use of this method since that time, and must say that, with hematite ores at least, I have found it more rapid than accurate. In some of the soft blue hematites of northern Michigan, the phosphorus exists entirely as apatite, and is therefore entirely soluble in dilute nitric acid; in others, of exactly the same appearance, nitric acid fails to extract all the phosphorus, which is easily dissolved by hydrochloric acid; and in still other ores, I have found that without previous fusion with alkali carbonate neither nitric nor hydrochloric acid will extract all the phosphorus. As it is impos-

sible to tell from the appearance of an ore whether all or part of the phosphorus is soluble in nitric acid, I have found this method of use only where rapid generalization of an ore as Bessemer or non-Bessemer was desired. Even in such cases its use was attended with a certain risk; for if low-phosphorus results were obtained, there was always a doubt whether the ore was really Bessemer, or whether its phosphorus was only slightly soluble in nitric acid. On the other hand, if high results were obtained, the ore was certainly non-Bessemer. For such purposes, I have used this method in connection with permanganate titration of the yellow precipitate, and have obtained results afterwards found to be accurate within 0.005 per cent. in three hours after the sample was taken from underground. For the analysis of certain ores, as, for instance, that from the Vermilion mine, and from Lake Angeline, this method cannot be used, as the phosphorus is not entirely soluble, either in nitric or hydrochloric acid, and such ores must be fused with alkaline carbonates in order to render the phosphorus soluble.

In the majority of hematites, almost all the phosphorus is extracted by one hour's boiling with dilute nitric acid. Especially is this the case with non-Bessemer ores. For example, in a hematite containing 0.507 per cent. phosphorus, on boiling 2 grammes with 50 c.c. nitric acid (sp. gr. 1.10), I found, after one hour, 0.501, and after an hour and a half, 0.503 per cent. of phosphorus had been removed. In lower ores the percentage of phosphorus dissolved was not so great. From an ore containing 0.063 phosphorus, by boiling one hour with 100 c.c. nitric acid, I found that 0.059 phosphorus was dissolved. These examples show close agreement, but such results are not always obtained. In rapid work on samples from underground, where accurate results are desired as soon as possible, I have found Emmerton's volumetric method invaluable; but for stock-pile and car-samples, and for analyses of ore sold under close guarantee, I have found nothing superior to the old-fashioned method of estimation as magnesium pyrophosphate.

MR. WOODBRIDGE (Communication to the Secretary): I have read with pleasure Mr. Browne's communication in regard to this method, especially as he gives his experience with hematite, while my experience has been mainly with magnetite. In the few hematites I have examined I have had but one failure; and in that case the sample was too small for a thorough test. From magnetite I have succeeded in every experiment in extracting all the phosphorus in one half-hour's boiling with the dilute acid (one part nitric acid

in four parts water). As indicated in the title of my paper, no claim of accuracy is made for this method, excepting in the case of Mineville magnetic ore; and, although Mr. Browne's experience indicates that it is not safe to apply this method *generally*, still, if care is taken to test its accuracy thoroughly on ore from a certain locality, it may be used with great advantage. As an instance, I have made six phosphorus determinations in three hours in ores carrying from 0.02 to 0.07 per cent. of phosphorus, ten grammes of each sample being taken for analysis. I agree with Mr. Browne that, at the present time, the court of final resort in a phosphorus determination is the "magnesia method;" and, therefore, would not recommend the dilute nitric acid method, except for general work and with occasional verification of results.

SUPPLEMENTARY NOTE ON BLAST-FURNACE LINES.

BY EDWARD WALSH, JR., ST. LOUIS, MO.

(New York Meeting, February, 1880.)

THE difficulty of securing for experimental research the actual conditions to be found in practice very frequently deters many from engaging in such work. Probably no metallurgical operation is more difficult to imitate in the laboratory than the blast-furnace process. So many complex reactions take place within the furnace, under such varied conditions of time and temperature, with innumerable mechanical, physical, and chemical mutations, that while the results reached may approximate those obtained in practice, there is always something wanting. Much work of this kind has, nevertheless, been done, and is now being undertaken. We are indebted to the bold minds who have attempted it for our present real knowledge of the blast-furnace process. Not that those who have performed series of experiments at a few given temperatures can claim the honor. It required more than this—the ability to make a summary of all preceding experiments, to supply those which were wanting, and to confirm those that were doubtful.

Sir I. Lowthian Bell, whose able papers before the Chemical Society and the British Iron and Steel Institute (forming the basis of his great work, *The Chemical Phenomena of Iron-Smelting*) has

contributed more than anybody else to an exact chemical knowledge of the blast-furnace.

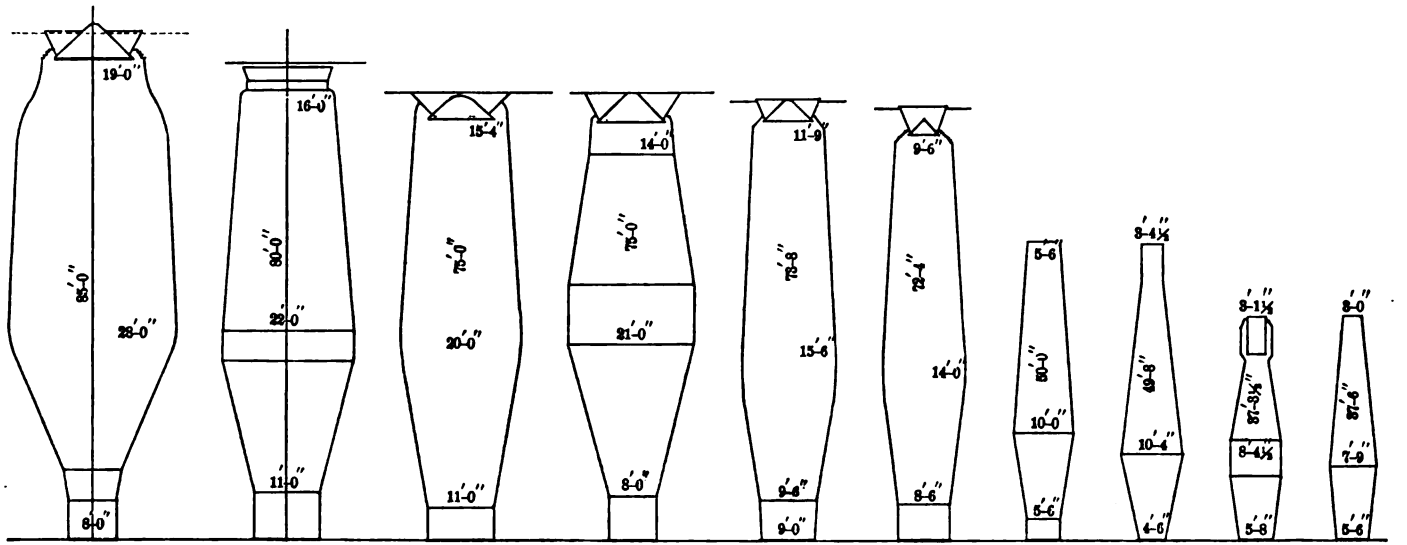
To the practical man the knowledge of the chemistry of the furnace has advanced far enough to meet present requirements, for practice has now fully availed itself of its teachings. For technical *finesse* there is, however, very much yet to be determined. It has been clearly demonstrated that carbonic oxide (with the necessary appliances for furnishing and heating the air and with proper admixtures of ores, fuel, and stone) is the main factor in the blast-furnace process. The generation of this gas before the tuyeres furnishes the heat to melt the slag and metal, while the gas itself is the reducing agent throughout the entire process. It reduces iron oxides into metal, and in the cooler portions of the furnace it furnishes by dissociation the metalloid carbon, in a more attenuated form than in the solid fuel, to form fusible cast-iron. The temperatures at which this gas acts on iron-ores have been shown, while the temperatures at which it fails to exert a reducing influence have also been determined.

With this information before us, we may reasonably consider the main requirements of the blast-furnace to be: First, that it should be a good generator of carbonic oxide gas; Secondly, that the gas thus produced should be used as effectively as possible on the materials to be treated.

In a former paper on the "Irregularities of the Blast-Furnace Process, and a Practical Way to Avoid Them,"* I attempted to show that "by placing the bosh or widest diameter of the furnace entirely and safely within the zone of complete fusion," "a constant supply of fuel before the tuyeres" would be secured. Under this condition (with the aid of a reliable blowing-engine) the furnace becomes, within the fusion-zone, an excellent gas-producer, thereby, as it would seem, entirely meeting the first requirement.

This suggestion was received, as I afterwards learned from various sources, as worthy of consideration. In the discussion of the paper, however, I was severely criticized for attempting to suggest a reduction of cubical capacity with the object of gaining sure mechanical delivery of the solids, and obtaining high reducing effects by restricting the sectional areas of the zones in the shaft. My aim was to secure a greater tension of the reducing gases, and a better distribution of them throughout the furnace-shaft, thus to promote their

*Trans., xv., 419.



Scale: $\frac{1}{32} = 1'$

Name of furnace.	Fuel.	Cubic Capacity. Cubic feet.	Total Height. Feet.	Diameter of Hearth. Feet.	Diameter of bosh. Feet.	Diameter of Throat. Feet.	Daily Product. Tons.	Time in Furnace. Hours.	Fuel per ton of Metal. Pounds.	Location.	Authority.
1. Newport	Coke	30,000	85′	8′	28′	19′	70	150	2308	Newport, Eng.	Sir B. Samuelson.
2. Edgar Thomson "F"	"	18,000	80′	11′	22′	16′	270.9	21	1863	Pittsburgh, U. S.	Mr. James Gayley.
3. N. Chicago, No. 6	"	15,000	75′	11′	20′	15′4″	175.4	24	2116	Chicago, U. S.	Mr. E. C. Potter.
4. N. Lonsdale, No. 3	"	14,000	75′	8′	21′	14′	107.1	36	2130	Lancashire, Eng.	Mr. Edmund Tosh.
5. Union, No. 3 and No. 4	"	8,824	73′8″	9′6″	15′6″	11′9″	142	20	1943	Chicago, U. S.	Mr. C. H. Foote.
6. Union, No. 2	"	6,676	72′4″	8′6″	14′	9′6″	114.3	19	1697	Chicago, U. S.	Mr. C. H. Foote.
7. Midland	Charcoal	2,000	50′	5′6″	10′	5′6″	48	8	1870	Crawford Co., Mo., U. S.	Mr. Wm. H. Lee.
8. Treibach, No. 3	"	1,872	49′8″	4′6″	10′4″	3′4½″	24	10	1365	Treibach, Austria.	M. Edouard Gruner.
9. Ferdinand	"	1,235	37′3½″	5′8″	8′4½″	3′1½″	21	4	1653	Hieflau, Austria.	M. Edouard Gruner.
10. Wrbna	"	1,000	37′6″	5′6″	7′	3′	20	7	1585	Eisenerz, Austria.	Sir I. Lowthian Bell.

Lines of Ten Blast-Furnaces, with Comparative Data of their Dimensions and Work.

better transpiration through the ores; thereby adding largely to the reducing effect, as announced.

The statement of my original paper, "Hence we may conclude that the more narrow, within practical limits, the furnace-shaft is constructed, the more thorough and the more energetic the actions of reduction and carbon-impregnation will be," etc., was deliberately made. I have failed to observe anything since to make me alter the practical conclusion there reached.

It was drawn rather from comparisons of the actual workings of furnaces of different designs than from the tests of Mr. Bell, which, however, tend to confirm it.

To show that the above views are correct, I submit the outlines of a few furnaces with comparative data of their work in tabular form, as shown on page 756. It may be well to add that the Union, No. 2 (Fig. 6), and the Treibach, No. 3 (Fig. 8) (which require less coke and charcoal respectively per ton than any furnaces that have come under my observation), do not owe their low fuel-consumption to large cubical capacity, to the employment of high temperature of blast, or to the great length of time the materials are allowed to remain in the furnaces.

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*A RAPID METHOD FOR THE REDUCTION OF FERRIC SULPHATE IN VOLUMETRIC ANALYSIS.**

BY CLEMENS JONES, HOKENDAUQUA, PA.

(New York Meeting, February, 1889.)

Discussion.

J. K. Eveleth, Anaconda, Montana (Communication to the Secretary): Partly for the reasons which Mr. Jones enumerates, and partly because of the presence of copper in greater or less quantities in so large a proportion of the ores throughout this region, I have also found Marguerite's method the most convenient and satisfactory for the determination of iron. In order to avoid the previous precipitation of the copper with H_2S on the separation of the iron by means of ammonia, with the consequent necessity for re-precipitation

* See page 411.

and repeated washings, in case the percentage of iron is at all large, my usual practice has been to introduce the zinc, by means of a platinum basket, into the solution containing the sulphates of iron and copper. De-oxidation being completed, the solution of ferrous sulphate is filtered from the precipitated copper into a titrating flask, in which is hung another basket containing two or three small pieces of fresh zinc, by means of which a gentle action is kept up, just sufficient again to reduce any iron which may possibly have been re-oxidized during the process of filtration; and in a few moments the solution is ready for titration. By this use of the baskets, the operation can be hastened considerably, since the zinc can be withdrawn as soon as the reduction is complete. Moreover, this does away with the necessity for the application of heat and the addition of a large amount of sulphuric acid, in order to dissolve such zinc as may be in excess. At the best, however, this process requires considerably more time than Mr. Jones finds necessary in the use of his ingenious apparatus. But it would seem that, unless a previous separation of iron and copper is effected before reduction, the precipitated metallic copper might interfere considerably with the rapidity of the operation, long before the exhaustion of the pulverized zinc in the tube.

MR. JONES (Communication to the Secretary): As Mr. Eveleth observes, the presence of a large amount of copper would interfere with rapid titration, in reducing ferric sulphate solutions. Still, a few moments only would be required to replace the pulverized zinc for each filtration through the reductor, so that, with strong suction, a ferric solution could be reduced in ten minutes at most. Or, an arrangement of several zinc plates, similar to a Smee galvanic cell, lowered into the solution, would precipitate the copper in three or four minutes. The plates could be washed off, and the solution filtered, and then transferred to the reductor.

THE REOPENING OF THE TILLY FOSTER IRON-MINE.

BY F. H. McDOWELL, NEW YORK.

(New York Meeting, February, 1889.)

DURING the last eighteen months the Tilly Foster Iron Mine Company, the President of which is Mr. E. F. Hatfield and the Managing Director Mr. B. G. Clarke, has been carrying out a plan

for the reopening of its iron-mine at Tilly Foster, Putnam county, New York, which, for boldness of design and promptness of execution, stands unrivalled.

The credit for bringing the operation to its present stage of success belongs to Mr. E. S. Moffat, general manager, who has had charge of the whole operation, and Mr. Clinton Stephens, of the firm of Stephens & Arnold, contractors. The writer has been connected with the work as engineer.

An account of the mine and its history will be found in the paper of Mr. A. F. Wendt, on "The Iron-Ores of Putnam County, New York" (*Trans.*, xiii., 478); and a very able and interesting discussion of the structure and form of the deposit is contained in the paper of Mr. Ferdinand S. Ruttmann, "Notes on the Geology of the Tilly Foster Ore-body" (*Trans.*, xv., 79).

The Tilly Foster mine, before its purchase by the present owners, was worked in a desultory way, whenever the price of ore reached the point of a small profit over the high rates of freight and cost of mining. Under the present management, a larger and more regular output was maintained until the old system of mining could no longer be pursued. The old system consisted in sinking on the ore-body from the surface to the 165-foot level, and leaving ore-pillars to support the hanging-wall, the vein being over 100 feet wide at this level, and the overhang, in places, nearly 50 feet.

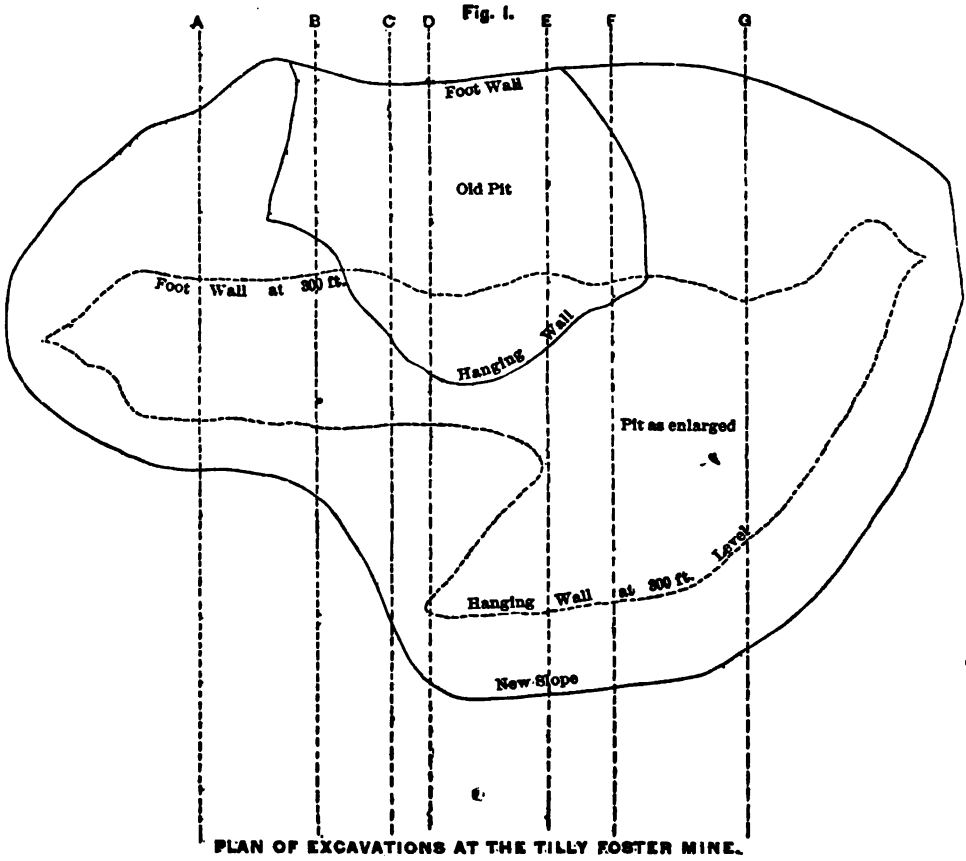
When these pillars proved inadequate and caves occurred, both ore and rock were removed from the pit, and the ore was assorted on the bank, precautions being taken to prevent, by the erection of dry masonry and cement walls, the spread of these caves at the ends of the pit.

After the pit was exhausted, new workings were opened below the 165-foot level by means of inclines sunk on the foot-wall, which has a slope of about 66°. Stations were cut and drifts were run right and left along the foot-wall at every 100 feet in depth, and cross-cuts were made to the hanging-wall, with upraises into chambers, 20 feet wide, leaving pillars 20 feet thick and floors from 15 to 25 feet hick. Then an effort was made to rob the mine of its pillars, first by springing brick arches at the south end from foot- to hanging-wall, to take the place of the pillars, and later, by drawing the ore from the chambers after caves had been developed in both floors and pillars. These efforts failed, as did, in turn, every other scheme devised for the extraction of the reserves. The situation called for heroic measures; and the plan finally adopted necessitated the hand-

ling of over 500,000 tons of rock, with the expenditure of more than $250,000.

Fig. 1 is a plan, and Figs. 2, 3, and 4 are sections selected from fifty, taken 10 feet apart throughout the length of the deposit. It will be seen that the scheme adopted necessarily involved stripping to the 165-foot level at all points. In some parts of the mine, where the greatest width of ore-body occurs (as shown for instance in Fig. 5), the stripping must go even deeper. The new hanging-

Fig. 1.



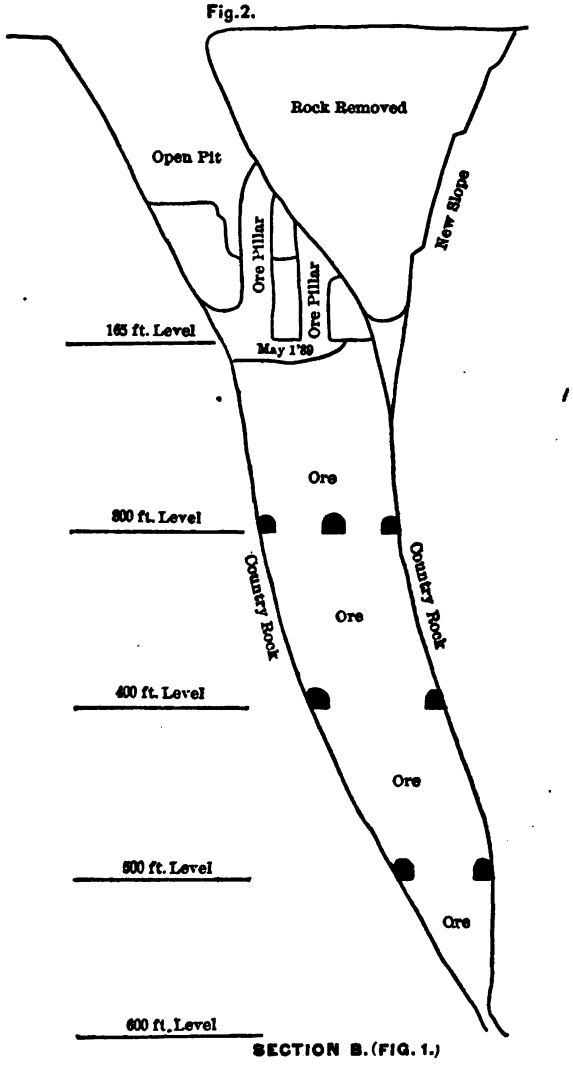
PLAN OF EXCAVATIONS AT THE TILLY FOSTER MINE.

wall slope varies from a vertical position to an inclination of one foot horizontal in six feet vertical.

Up to the present time, no difficulty has been experienced in securing good, strong, natural walls, and the work of stripping is now (May 1st, 1889) so far advanced that Messrs. Stephens & Arnold have entered into a contract to deliver a minimum of 50,000 tons of shipping-ore during 1889.

The methods devised by Mr. Stephens for the rapid removal of rock and ore are worthy of consideration. At the narrow ends of the ore-body, steam derricks are in use, and across the intervening space of about 300 feet, four cables have been stretched. Two of

Fig. 2.

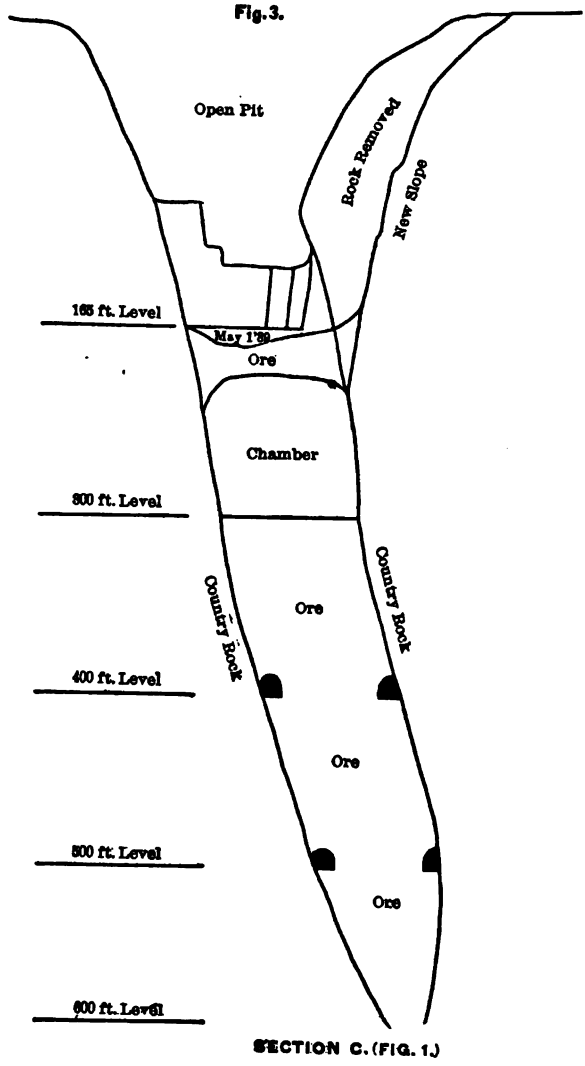


SECTION B. (FIG. 1.)

these, shown in Fig. 6, are horizontal cables, 2 inches in diameter, supported at each end by well-braced towers, and strongly anchored into the rocks. On each cable a trolley is moved back and forth

by a traversing rope, passing around one drum of the engine, while another drum of equal diameter carries the fall-rope which hoists the rock and ore out of the pit. As the engines are combined "link-motion" and "friction," the engineer can move his load horizontally or vertically at will. The other two cables are inclined,

Fig. 3.

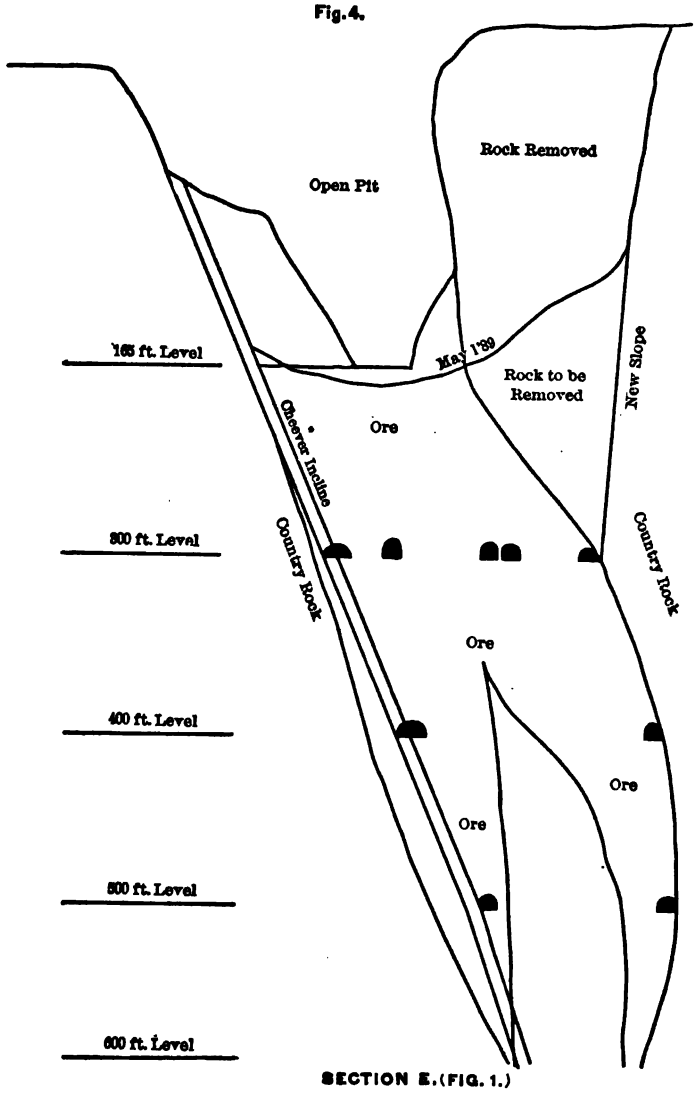


SECTION C. (FIG. 1.)

one end being anchored in the foot-wall about 100 feet from the surface, and the other, after passing over a tower, on the bank. In this system the movement of the trolley in its descent is regulated

by a stop-block, the location of which on the cable is controlled through a tail-rope by the engineer, one drum being made to raise the load vertically out of the pit and along the incline-cable to the surface.

Fig. 4.



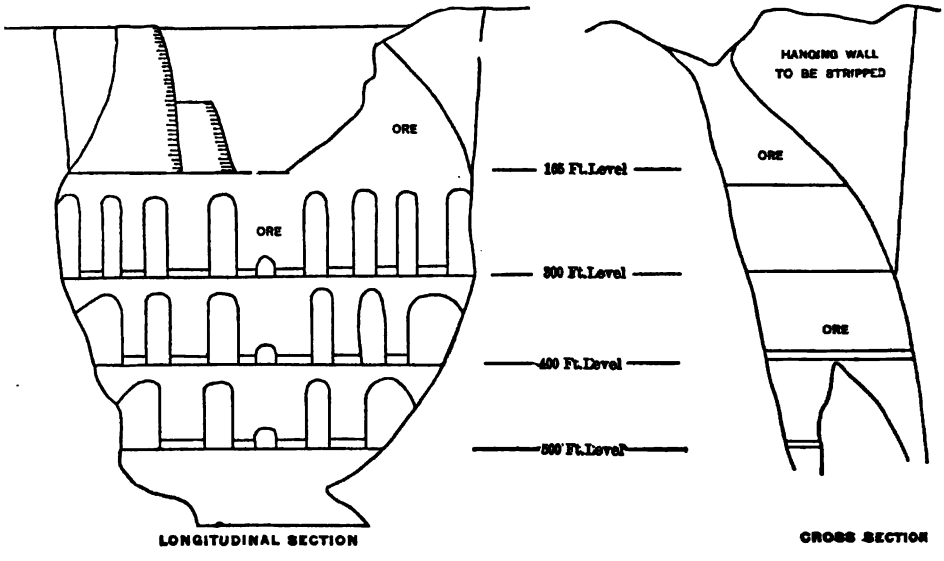
SECTION E. (FIG. 1.)

To save time and handling, the car-bodies are lifted from the trucks, lowered into the pit, and exchanged for loaded ones, which are hoisted to the surface, lowered upon the trucks, and run out to the dumps by horses. In this way 1000 tons of material have been

handled every ten hours. To expedite the work of loading the car-bodies, light auxiliary cables are used, by means of which pieces of rock too large for the men to lift are readily hoisted into the bodies, while the main cables are kept busily employed in removing those already loaded.

The ore has, in the past, been cobbed and assorted by hand on the bank. A large Brennan crusher has lately been erected, and hereafter the sorting will be done in the pit and the ore will be reduced to furnace-size on the bank. The taking of monthly measurements upon which to base estimates for contractors has been no small

Fig. 5.

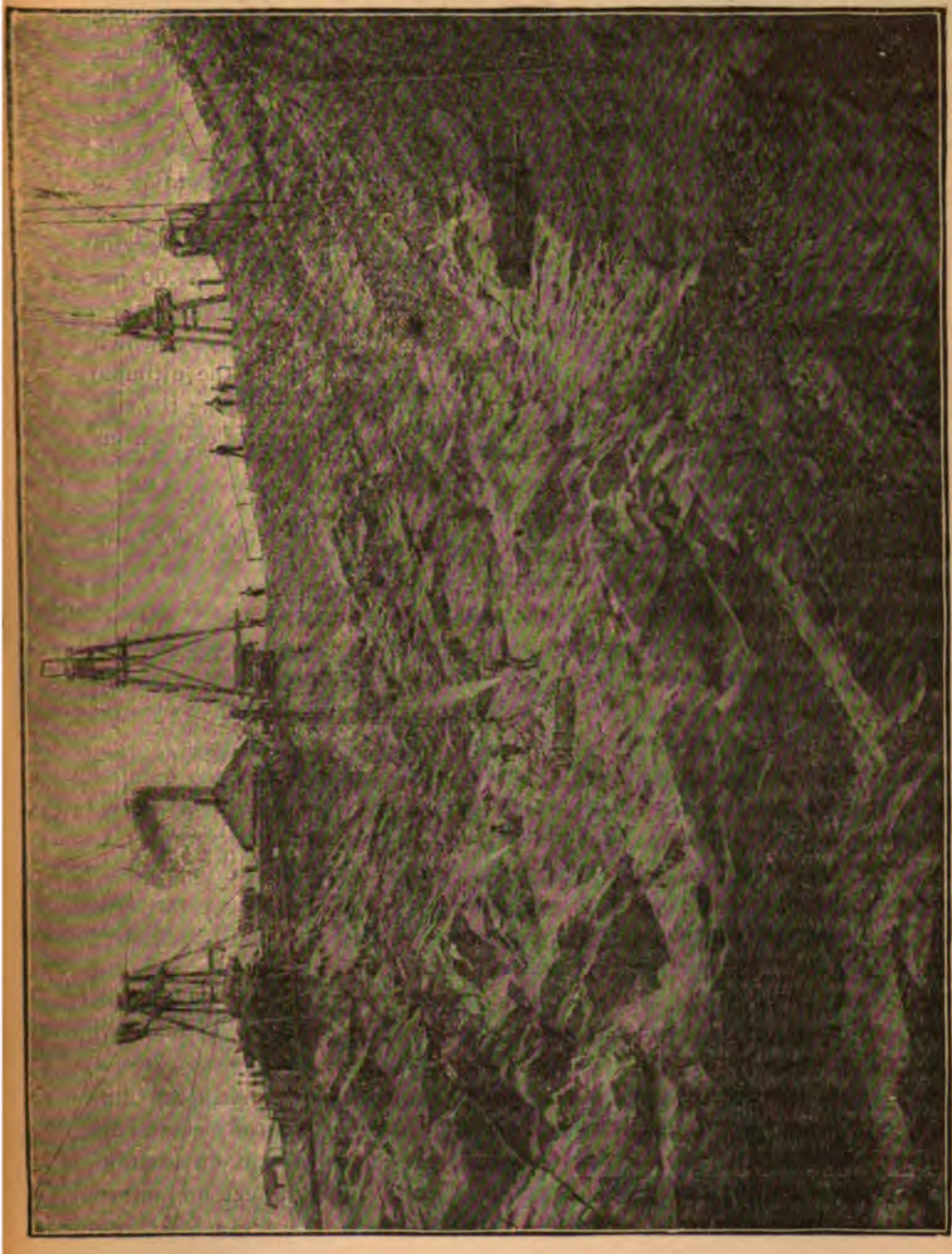


LONGITUDINAL SECTION THROUGH THE TILLY FOSTER MINE, AND CROSS SECTION AT THE WIDEST PART OF THE ORE-BODY.

task, especially as there are two classes of materials to be blasted and hoisted from each of the four levels, besides a special class, making nine classes in all, with their corresponding prices.

Fifty cross-section sheets, made up from careful preliminary surveys, upon which the elevations, taken every 10 feet across the pit, have been plotted, afford a means of keeping accurate records of the progress of the work. The planimeter has been used to good purpose at the end of each month to get up prompt returns for the contractors; and these have been checked, in turn, by prism-measurements, which are final and conclusive. At the writer's request, Mr. C. C. Mattes, the mining engineer of the Company, has

been associated with him in this work, and has lately developed a very comprehensive system of measuring the pit by means of a trol-



METHOD OF MOVING ROCK AND ORE FROM THE OPEN WORKINGS AT THE TILLY FOSTER MINE.

ley, traveling upon a light, suspended, movable cable, the leveler remaining with his instrument on the bank. This will prove invaluable as depth increases, and is worthy of a special paper.

Some idea of the length and height of the ore-pillars may be obtained by reference to the cross-sections made by Mr. Ruttmann, and shown in the illustrations of his paper already cited. The shipping-ore is now mined by the contractors for from 85 cents to $1.00 per ton, the lean ore being delivered to the dump at rock-prices, which are from $1.15 to $1.45 per cubic yard, according to the level hoisted from.

The undertaking has been based upon the expected recovery of 600,000 tons of shipping-ore, against which a royalty of $1.00 per ton is charged to cover the stripping and incidental expenses.

Earnest efforts are now being made to utilize the lean ore, which has heretofore been a waste product, and to increase the ultimate output of ore by judicious mining, in the hope of reducing the cost of stripping to 50 or 60 cents per ton on the aggregate of ore thus made available.

The following statement gives a comprehensive view of the progress of the work. Readers of the papers of Messrs. Wendt and Ruttmann will be aware that the rock is gneiss and the ore magnetite:

Number of Cubic Yards Removed.

June,	1887,	600.00	August,	1888,	14,858.41
July,	"	3,682.00	September,	"	13,850.75
August,	"	4,817.13	October,	"	12,896.66
September,	"	7,684.68	November,	"	12,669.97
October,	"	8,785.00	December,	"	12,024.95
November,	"	10,512.47	January,	1889,	11,939.07
December,	"	11,428.12	February,	"	10,309,18
January,	1888,	9,983.40	March,	"	12,149.44
February,	"	10,088.24	April,	"	7,406.87
March,	"	9,312.16			234,570.39
April,	"	10,707.18	Stripping-ore,		7,211.66
May,	"	13,221.78			
June,	"	13,443.95	Rock,		227,358 73
July,	"	12,198.98			

The ore removed during the progress of the stripping has amounted to 7211.66, leaving 227,358.73 cubic yards of rock. This amount of rock includes several thousand tons of lean ore accumulated on the dump, in addition to about 100,000 of such ore, mined

before the stripping began. The whole quantity is now awaiting the results of experiments in progress, to determine the details of subsequent treatment. At the present time the material hoisted is almost wholly first-class and lean ore.

THE OCCURRENCE AND TREATMENT OF THE ARGENTIFEROUS MANGANESE ORES OF TOMBSTONE DISTRICT, ARIZONA.

BY CHARLES W. GOODALE, BUTTE CITY, MONTANA.

(Utah and Montana Meeting, July, 1887.)

THE attention of the Institute has been called by Prof. John A. Church* and Mr. W. Lawrence Austin† to the free-milling ores of the Tombstone mines and their treatment, but the silver-bearing manganese ores of the district have received only a passing notice.

As these ores show some interesting peculiarities, and have yielded not less than 750,000 ounces of silver, they are worthy of some study.

The Knoxville, Lucky Cuss, Luck Sure, and Wedge mines (named in the order of their importance) are situated along the northern border of the district. They have been the leading producers of this manganese ore, and possess the same general characteristics. Special attention will be given to the Knoxville, as it has been the one most extensively developed.

The limestone belt, in which the Knoxville ore bodies occur, has an easterly and westerly trend, is about 1680 feet wide, and rests on granite on the north, the plane of contact being nearly vertical.

It is overlain by quartzite, the contact-plane having a dip of 82°. Some mining has been done along these contacts, and a considerable amount of pay ore has been extracted, but development has been very superficial. The ore found along these contact planes contains very little manganese, and differs entirely from the ore in the limestones.

No attempt will be made to assign this limestone country-rock to a particular geological horizon, as no fossils have been found in it.

Prof. Blake, in a paper on "The Geology and Veins of Tombstone, Arizona" (*Trans.*, x., 334), speaks of the middle and upper strati-

* Concentration and Smelting at Tombstone, Arizona. *Trans.*, xv., 601.

† Silver Milling in Arizona. *Trans.*, xi., 91.

fied beds of limestones, shales, and quartzites as "Palæozoic and probably lower Carboniferous." There are no positive evidences of stratification in the Knoxville limestone, although indications may be noted in some places which would lead to the conclusion that the strata are parallel to the plane of contact with the quartzite.

In cross-cutting a great variation in the character of the limestone is observed. Some zones are so siliceous as to approach quartzite, though the greater part of the rock is limestone, carrying 94½ per cent. of carbonate of lime.

The ore-chimneys, which dip to the east at an angle of 40° to 50°, occur along a crack or plane of cleavage nearly vertical, and having an easterly and westerly strike. Marked evidences of slips and faults are entirely wanting, and an effort to apply the theory of hot springs to account for the origin of the ore-bodies is met with the difficulty of explaining how a sufficient opening was made for the hot water to obtain access and cut out the large channels. The chimneys are somewhat irregular in shape and vary in size. Four of them, all more or less connected near the surface, have been developed—three to a vertical depth of 400 feet, and the fourth to 150 feet.

In drifting from the main shaft, the crack or plane of cleavage above mentioned was a sure guide to the ore-bodies, though its width was not appreciable, and there was not even a knife-blade seam of "clay" or "talc" between the walls.

There were no indications of ore until the drifts were within a few feet of the ore-bodies, where the walls of the crack were stained with black oxides of manganese and some carbonate. Small detached pockets of pure manganese oxide also indicated proximity to the chimneys, but these small bodies carried very little silver. The filling of the chimneys included, in a great variety of forms, pyrolusite, wad, and psilomelane. It is not improbable that a searching examination would have discovered the rarer oxides, braunite, manganite, and hausmannite. The gangue was quartz and calcite. Galenite, cerussite, pyromorphite, cuprite, melaconite, and malachite were occasionally observed, and the assay showed a little gold—about $\frac{1}{100}$ of an ounce, or 20 cents to the ton.

Caverns were found in the widest parts of the ore-bodies which were lined with snow-white and crystalline calcite. The purest manganese ore formed the lining of the chimneys, the percentage of gangue being greater in the middle. Two classes were therefore made of the ore in preparing it for shipment, the first class being

composed of the more siliceous and higher grade ore, and the second, being low in silica, was valuable as a flux, though some of it did not carry enough silver to pay for smelting.

The following analyses will show the character of the two classes of ore, which will be designated as "milling" and "smelting" ore:

Milling Ore.

Analysts: D. Baker, A. H. Low, M. W. Iles.

	Per cent.	Per cent.	Per cent.
Mn_2O_3,	74.16	42.77	41.30
SiO_2,	18.10	25.28	24.25
CaO,	1.32	21.60	18.75
CO_2,	1.33		
Fe_2O_3,	.58	3.54	6.80
MgO,	.13		
Na and K,	1.22		
Sb,	.66		
PbO,	.50		
CuO,	.20		
Ag,	.15		
Cl,	.17		
S,	.08		
H_2O,	1.67		
	100.27		

Smelting Ore.

Analyst: Walter Bunce.

	Per cent.
Mn,	47.70
O in combination,	26.22
SiO_2,	8.70
Fe_2O_3,	1.20
$CaCO_3$,	6.30
Cu,	1.21
Pb,	.45
Ag,	.06
H_2O,	6.00

The analyses of Messrs. Low and Iles were made from samples of mill-runs taken out within 100 feet of the surface. It will be observed that the percentages of Fe_2O_3 and $CaCO_3$ are much higher in their analyses than in the results obtained by Messrs. Baker and Bunce on ore taken out from a depth of nearly 350 feet.

All of the chimneys yielded ore assaying from 30 to 50 ounces silver per ton near the surface; but below 150 feet the silver con-

tents decreased as depth was gained, with the exception of No. 2 chimney (numbering from the left), which produced a uniform quality of ore so far as it was worked. It is therefore probable that the silver was brought up in solution through the main chimney (No. 2) and that the other chimneys received their silver by overflow from it.

It is probable that the chimneys were cut out by carbonic acid waters, for the walls show unmistakable signs of corrosive action; just the effect which would be seen on a piece of siliceous limestone after treating it with acids.

The origin of the manganese is worthy of study. Prof. W. P. Blake, after a hasty visit to the Knoxville, expressed the belief that the manganese had been segregated from the limestone, through which it had been originally disseminated as carbonate. After making an analysis of some pink-stained limestone near one of the chimneys, and finding much less manganese than he expected, he wrote: "This leaves the origin of the abundant black oxides more in doubt, but I still believe that the limestone is the source." A determination of manganese in a piece of limestone taken from the 340-foot level, about midway between chimneys 2 and 3, showed 0.1 per cent. manganese.

There is hardly sufficient evidence to justify the belief that the manganese was originally formed in the chimneys as carbonate and silicate. If, on following these chimneys down below water-level—perhaps 300 feet deeper—the fillings were found to be rhodochrosite and rhodonite below all decomposing influences, all doubts as to the first form of deposition would be settled. It would be much easier to account for the origin of these highly oxidized ores if they were found in bedded deposits, for then the theories which apply to the deposition of hematites and limonites might explain their origin.

The treatment of these argentiferous manganese ores, with no smelting works in the district and no market for them nearer than Colorado or San Francisco, was the problem to be solved in Tombstone in the fall of 1881.

Efforts were first made to work them by free milling, the heaviest manganese being sorted out before shipment to the mill. This method was soon abandoned, as the silver saved did not average 60 per cent., and the loss of quicksilver was excessive—not less than 7 pounds to the ton of ore. Owing to the presence of so much manganese, this loss of quicksilver was to be expected; but it is not known whether the low yield can be accounted for by the failure to

amalgamate or by the "flouring" of the amalgam with the quicksilver. Applying the test by hyposulphite of sodium to determine the amount of silver in the condition of chloride, it was found that only 24 per cent. of the silver was soluble in the hyposulphite. But this test, when tried on the best free-milling ores of Tombstone district, gave no indication of the yield to be expected by amalgamation. Such ores as the Vizina, Ingersoll, and Bradshaw yielded from 80 to 90 per cent. of their silver by free milling, but the hyposulphite chlorination test showed that from 45 to 84 per cent. of their silver was in the form of chloride.

A sample lot of the Knoxville ore was sent to the Massachusetts Institute of Technology for examination, and from tests and analyses made there, it was believed that the silver not combined with chloride was united with copper, antimony and sulphur in tetrahedrite. It will be seen from the analyses already given that no tellurium was reported, though it is not improbable that this element was present combined with silver and gold. In Prof. Church's paper (*Trans.*, xv., 602), the presence of tellurium is mentioned in the ores mined by the Tombstone Mining and Milling Company and it was found in other mines of the district.

Before giving up the hope that some of these manganese ores could be treated by the free-milling process, every chemical known in the business was tried, and in varying quantities, but it seemed impossible to prevent the manganese from fouling the quicksilver and causing a great loss. The bullion product was nearly pure silver, and so strong was the oxidizing power of the manganese oxides that the bullion showed no trace of copper even when, as an experiment, a pan-charge was run with one hundred pounds of bluestone.

After some experimental work in chloridizing-roasting in a small reverberatory furnace, under the direction of Mr. John H. Collier, which proved that the ore could be chloridized and amalgamated up to 86 per cent., the Boston and Arizona Smelting and Reduction Coompany added a White and Howell furnace and revolving dryer to their 20-stamp mill-plant, arranged 15 stamps for dry crushing, and began the treatment of the higher grade of Knoxville ores by this process. It was found by experiments with higher and lower percentages of salt that the amount required for the proper chloridization of the silver was 6 per cent. of the weight of the ore. Ore and salt were shoveled into the rock-breaker together. Dropping into the dryer, and from the dryer into the self-feeders, it was ready for the

stamps. The batteries were provided with 40-mesh screens, it having been found that with a coarser screen the furnace could not do effective work. This showed that the silver was very intimately associated with the manganese. From the batteries the pulp was taken by the ordinary elevator and screw-conveyor system to the Howell furnace, which was 24 feet long and 4 feet in diameter. The cylinder was lined with fire-brick, and, acting on the idea that it was important to keep the ore from falling through the flame as much as possible, half of the bricks were placed edgeways. A short run, however, showed that the draft carried too much of the pulp into the dust-chamber, and the bricks on edge were chipped out. A marked improvement was the result, but after making another short run, a diaphragm or flange was placed on the upper end of the cylinder, which left the opening only 3 feet in diameter, and caused a still further reduction in the amount of pulp passing into the dust-chambers. This ore in the dust-chambers or flues would not have given much trouble if the auxiliary fire had been effective in completing the chloridizing action which was begun in the cylinder, but with this ore showing a chloridization of only 30 per cent., the auxiliary fire was given up as useless. There is no doubt that this dust could have been treated to the best advantage by taking it, by means of an elevator and screw-conveyor, to the lower end of the furnace and dropping it into the chamber which received the hot ore from the cylinder. But in order to carry out this plan so that the stream of ore would be regular and the action automatic, an entire reconstruction of the furnace would have been necessary. A different plan was therefore adopted. The dust-chambers were cleaned out every day, and the furnace-men fed the dust into an elevator, which raised it up to the main screw-conveyor carrying the ore from the batteries to the furnace. It may be surprising that it should stay in the furnace any better the second time it was put in than it did at first. The fact that it did so may be accounted for on the theory that the flue-dust possessed different characteristics, having been subjected to the heat of the dust-chambers long enough to lose its carbonic acid, oxygen, and moisture not expelled in the dryer.

These changes led to an increase in the chloridization from 85 up to 90 per cent. The tailings contained about 17 per cent. of the pulp-assay, which would leave 83 per cent. in bullion, but the bullion product was 81½ per cent. This difference is accounted for by the loss in dust, some of which collected on the roof of the waste-building. It assayed about 70 ounces per ton, when the pulp-assay was about 50 ounces.

An examination of the quicksilver in the pan-charges showed a somewhat broken condition, but it was no longer foul, as in the case of raw amalgamation, and there was little difficulty in collecting it in the settlers. However, it seemed impossible to avoid a considerable loss of quicksilver—about three pounds and a half to the ton of ore.

The experience of mill-men, and the theory of the reactions in a chloridizing-roast, demanded the addition of sulphur in some form to an ore like this which contained less than 0.1 per cent. of that element. (See *Roasting of Gold- and Silver-Ores*, by G. Küstel, pages 22 and 27.) But the result showed that the decomposition of the salt was vigorous enough, and that chlorination of the silver was effected without the presence of sulphuric acid. It might have been possible to get a higher chlorination of the silver if the theory of Küstel—that "calcareous ore requires as much more green vitriol or iron pyrites as is necessary to transform all the lime into sulphate"—had been put into practice; but sulphuretted ores in sufficient quantity were not available, and the use of copperas was out of the question, as will be seen by the following figures: The analyses of Knoxville ore show from 2.65 to 21.60 per cent. of $CaCO_3$. These were probably extremes, and the average was not far from 10 per cent. This would give 5.6 per cent of CaO, or 112 pounds to the ton, and in order to convert this into CaO, SO_3, 160 pounds of SO_3 would be required, or more than 500 pounds of copperas ($FeSO_4 + 7H_2O$).

It is evident that this method of introducing sulphuric acid would not do on 50-ounce ore, when the cost of copperas laid down in carload lots exceeded 4 cents per pound.

It is worthy of note that the ore as it dropped from the cylinder into the chamber was at a very low red heat, and imperfectly chloridized (not over 60 per cent. of the silver being combined with chlorine), but that while accumulating in the hot chamber the reactions went on with vigor.

The total production of the Knoxville mine to date has been 6000 tons of "milling" ore, averaging 43 ounces per ton, and 6200 tons of "smelting" ore, assaying 23.3 ounces per ton.

Nearly all of the smelting-ore was worked by the Tombstone Mining and Milling Company in smelting concentrates, and Prof. Church has mentioned the treatment of this ore in his paper already referred to on "Concentrating and Smelting at Tombstone."

If the Tombstone district had produced lead-ores so that the "milling" ores could have been worked by smelting, the profits of

the mine would have been greatly increased. Milling was too expensive with salt at $30 per ton, boiler-wood at $9, and pine wood for roasting at $12 per cord (making the cost per ton of ore for steam fuel $3.50, and for roasting $2).

I am indebted to Mr. Frank C. Earle for the following analyses of the limestones of the Tombstone district:

	$CaCO_3$.	$MgCO_3$.	SiO_2.
No. 1, . .	91.33 per cent.	trace	6.00 per cent.
No. 2, . .	90.75 "	2.85 per cent.	5.20 "
No. 3, . .	96.46 "	trace	2.41 "

No. 1. Main working shaft, "Lucky Cuss," near contact with granite. White and crystalline.

No. 2. West end of "Knoxville." Blue and compact.

No. 3. "Luck Sure." Represents general character of limestone in the belt.

What little Fe, Al and Mn occurred in the samples was not determined.

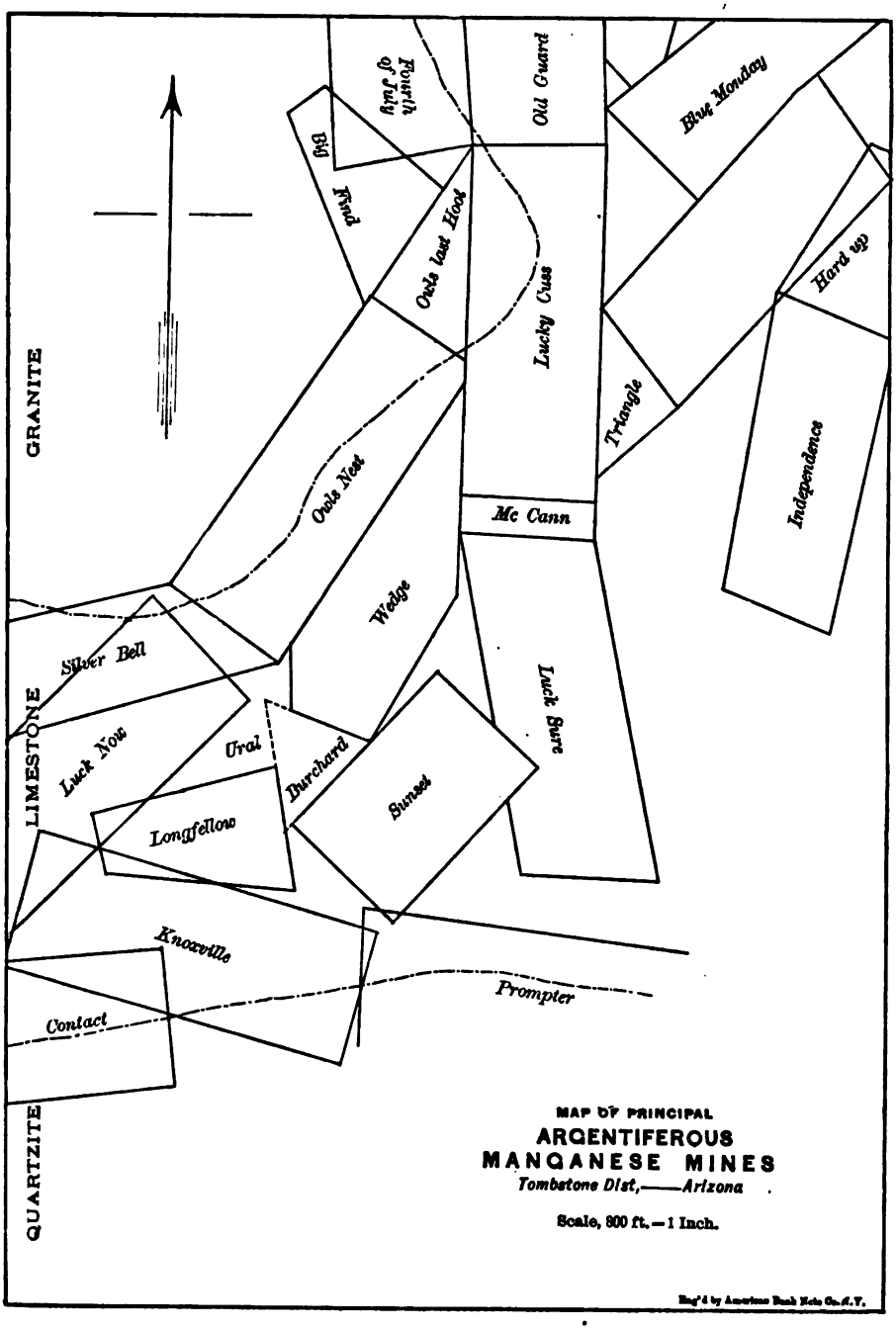
I am also indebted to Mr. H. G. Howe, M.E., for the accompanying map, showing the location of the group of mines referred to. The dotted lines indicate the granite-limestone and limestone-quartzite contacts as traced on the ground.

In the eastern part of Tombstone district there is another belt of limestone, in which are located the Bunker Hill, Rattlesnake, and Mammoth claims; the ores from these claims contain much manganese, though not to the same extent as the Knoxville, Lucky Cuss, and Wedge ores.

Discussion.

Richard Pearce, Argo, Colo. (Communication to the Secretary): I do not quite see the necessity for the "channel-theory" to account for the formation of these deposits. The gradual substitution of one mineral by another, through the agency of water and carbonic acid, would be sufficient, in my judgment, to account for them.

In connection with the remarks of Mr. Goodale as to the original form of the manganese minerals in these ore-bodies, I may observe that somewhat similar conditions exist at Butte, Montana, where large deposits of psilomelane and pyrolusite are found in comparatively isolated positions in the country-rock and are evidently the result of water-action. Carbonate of manganese enters largely into the composition of the veins adjoining these isolated deposits, and


GRANITE
LIMESTONE
QUARTZITE
Big Find
Fourth of July
Old Guard
Blue Monday
Owls last Hoot
Lucky Cuss
Hard up
Owls Nest
Triangle
Independence
Mc Cann
Wedge
Silver Bell
Luck Now
Ural
Burchard
Longfellow
Sunset
Luck Sure
Knoxville
Contact
Prompter
MAP OF PRINCIPAL
ARGENTIFEROUS
MANGANESE MINES
Tombstone Dist,——Arizona
Scale, 800 ft.—1 Inch.

water circulating through the veins, containing, perhaps, manganese carbonate in solution in carbonic acid, finds some lateral outlet through the natural joints of the rock, and by a process of gradual displacement these manganese oxides are formed.

Similar conditions also exist in regard to the silver contents of these Butte deposits. I think that, as a rule, the richer the ore in manganese, the poorer in silver; this would indicate that possibly silica may play some important part in the transfer of the silver from one point to another.

Mr. Goodale's account of mill-experiments confirms my impression that the action of MnO_2 on quicksilver in the amalgamation of silver-ores is scarcely understood. It appears from all the evidence we can obtain, that under certain conditions MnO_2 is capable of giving up its oxygen in some nascent form which acts most energetically on the quicksilver. This action is, in all probability, oxidizing, as the flouring of the quicksilver may generally be avoided by the addition of some baser metal, such as lead or copper, these metals being more readily oxidized than mercury. In the treatment of silver-ores, containing much base metal, by amalgamation, it is often found desirable to add manganese oxide, which has the effect of preventing the reduction and amalgamation of the base metals and leads to the production of a much finer quality of bullion.

Mr. Goodale speaks of the supposed necessity of adding sulphur to effect a chloridizing roasting. This my own practice leads me to think an error. I worked the Augustin process for many years on copper in a perfectly oxidized form, containing silver, without the aid of sulphur in any form. When the oxide of copper was brought to a proper temperature in the furnace, the chloridizing was effected by the addition of a mixture of fine siliceous sand and salt, moistened with water. The following is probably the reaction:

$$2NaCl + SiO_2 + H_2O = Na_2O, SiO_2 + 2HCl.$$

J. K. Clark, Butte City, Montana: I can add very little of interest to what has already been said about manganese-ores in milling; but it may be a point worth noting that it has been our experience at the Moulton mill that wherever a large amount of zinc has been present in our ores, the loss of silver by volatilization has been much reduced by a plentiful mixture of oxidized manganese ore.

Another point may be mentioned as showing the important action

of iron in assisting amalgamation. While running the Dexter mill in Butte, in 1876, where we chloridized in reverberatory furnaces, the tailings continued to rise in value most unaccountably, until the saving was less than 60 per cent., though the chlorination tests showed 95 and 96 per cent. of the silver in the form of chloride and proved that there was no fault in the furnace-work. This happened while we were expecting new mullers, and as soon as they arrived and were put in, the yield went right back to where it should be, according to the chlorination-tests.

An examination of the old mullers and other iron parts of the pan showed that they were so heavily coated with iron oxide, say $\frac{1}{16}$-inch thick, that no metallic iron could come in contact with the pan-charge. Since that time, I have taken good care that there is no lack of bright clean iron in the pans, and I believe the plan we have adopted at the Moulton—placing two or three wrought-iron bands, $\frac{1}{4}'' \times 4''$, around the inside of the pan—will always assure enough live iron to assist the amalgamation.

It is generally believed that the poor results obtained in a pan-charge of slimes alone are due to the thinness of the pulp, and the fact that the quicksilver cannot be well mixed with it and come in contact with the silver; but it is my opinion that the coating of the iron parts of the pan by the slimes, which would prevent the activity of the iron, may offer a better explanation of such results. When slimes and sand are worked together, the scouring action of the sand gives the iron a chance to play its part.

MR. GOODALE: In connection with the remarks of Mr. Pearce about the oxidizing action of MnO_2 on quicksilver, I would say that the Boston and Arizona Company purchased all the base and sulphuretted ores they could get, and offered good prices for such ores to mix with the Knoxville ore. It was found that whenever a mixture could be made which would yield a bullion-product containing from 5 to 10 per cent. of base metal, the saving was increased and the quicksilver was in better condition.

PROPOSED RAIL-SECTIONS.

BY ROBERT W. HUNT, CHICAGO, ILL.

(New York Meeting, February, 1889.)

WHEN I had the honor of presenting to the Institute at the Buffalo meeting in October, 1888 (*Trans.*, xvii., 226), my paper on "Steel Rails and Specifications for their Manufacture," I expressed my hesitancy in submitting my ideal rail-sections. Since then, continued residence in Chicago, perhaps, or greater intercourse with railroad officials, has either given me more courage or has taught me that so little is really known on the subject that one is safe in expressing one's views, let them be ever so erroneous. As a further personal excuse, I may remind you that "fools rush in," etc. At all events, I would now call attention to certain rail-sections which I believe to be good. I do not say they are the best, but I feel sure they have more merit than the majority of the adopted heavier-sectioned rails.

In determining the proper form for a rail, the subject should be considered from two points; first, and most important, the duty required; second, and of almost equal importance, the influence of existing details of manufacture upon the character of the finished product.

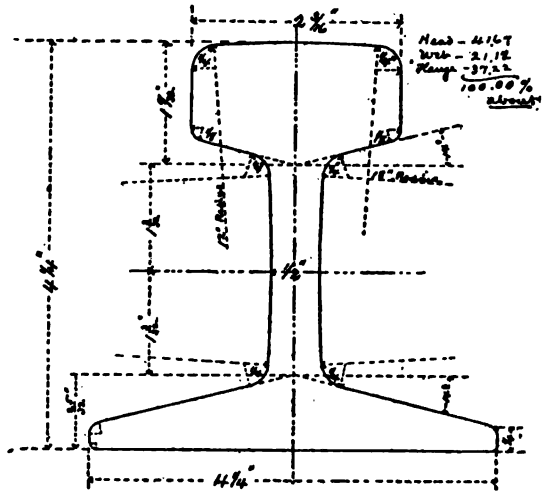
It is perfectly proper that all the strains to which the rail will be subjected should be considered and calculated. But its ability to resist them will depend quite as much upon the character of the metal as upon the form of its section. That which would be the realization of the ideal with *good* steel would be disappointment with poorer metal. Therefore, the railroad engineer should always have in mind and pay due respect to what the metal worker *may* give him—the *may* being the factor of danger to be covered by his factor of safety, and possessing greater importance than any formula laid down by never so eminent authorities. It is well-known how positive is my conviction that high heat during manufacture and good rails cannot go together; hence, if we insist upon the avoidance of this evil by rail-makers, we must give them sections which can be made at low temperatures with commercial as well as technical success.

The market-price for rails controls the practice of makers, and,

indirectly, the specifications of buyers also. Of course, railroad managers want the best possible article, but they must also have it, or its semblance, at least, as cheaply as their neighbors. Great reforms are generally inaugurated by bold spirits who disregard the controlling influences. In this case, I do not think any fanaticism or martyrdom is required, but believe all necessary and practicable perfection can be obtained without seriously antagonizing or disturbing existing commercial conditions.

In my former paper I quoted Mr. O. Chanute to demonstrate how great has been the increase of wheel-pressures. Since then Mr. D. J. Whittemore, Chief Engineer of the Chicago, Milwaukee and St. Paul

FIG. 1.



60 lbs.

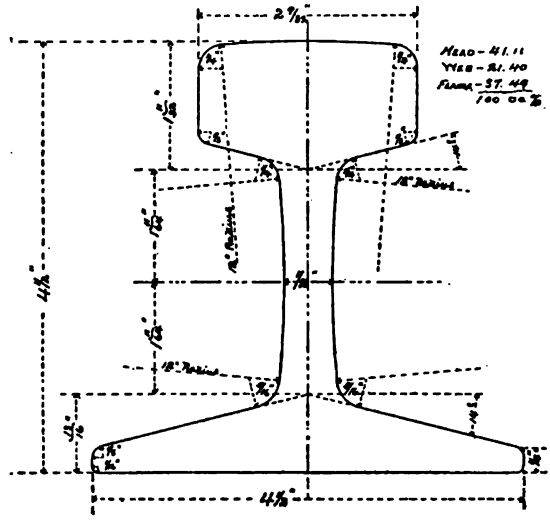
Railroad, has presented an able paper to the American Society of Civil Engineers, in which he further elaborates that point. I presume all authorities agree that the necessity exists for stiffer-sectioned or heavier rails. In other words, there is a demand for heavier rails, because the weight of wheel-pressure requires a stronger beam to support it. The railroad engineer naturally desires to secure this with the smallest outlay, and hence with the lightest practicable section. This can be designed, assuming, as before intimated, that the metal in it possess certain physical properties. But if the calculation is to be of value, these assumed properties must be present.

The metal commonly known as steel is almost as sensitive to

treatment as woman. It will stand any amount of punishment if administered by judicious hands, and unexpectedly rebel if a less amount is given with indifference. Now, we cannot always control the judgments, or rather the actions, of the irresponsible steel operatives; therefore sound policy dictates that we give them a section to which they can do the least injury.

The earlier rails which gave such satisfactory results, and which have been so constantly presented to rail-makers as representing that which *they* ought to do, were all of light sections. They were made of steel which had never been heated to the utmost limit of its endurance, and in mills in which the roll-pressures penetrated well through the rail-heads. In rolling later rails, the steel has been

FIG. 2.



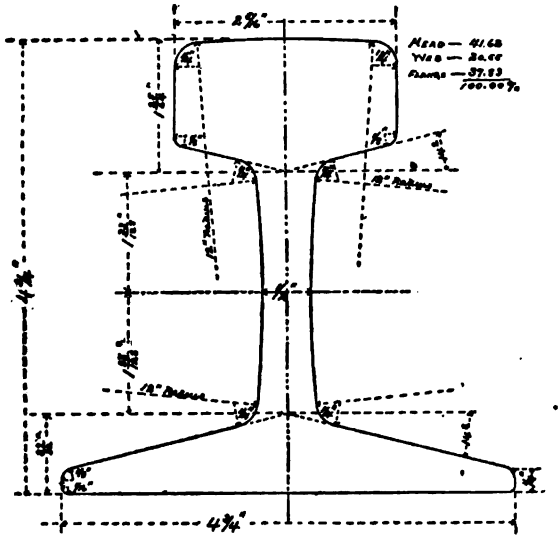
65 lbs.

almost always worked at a higher temperature; and, as the sections have been increased, the penetration of the roll-pressures has decreased.

It was supposed that the chemical character of the steel in the earlier rails accounted for their superior wear. Among the makers of these rails, Sir John Brown & Co., of Sheffield, England, sent to this country those which, from their most excellent service, were considered by railroad engineers as the type of what rails should be. Accepting the chemical theory, rail-makers expected that the analyses of these celebrated rails would present steel of exceptionable uniform-

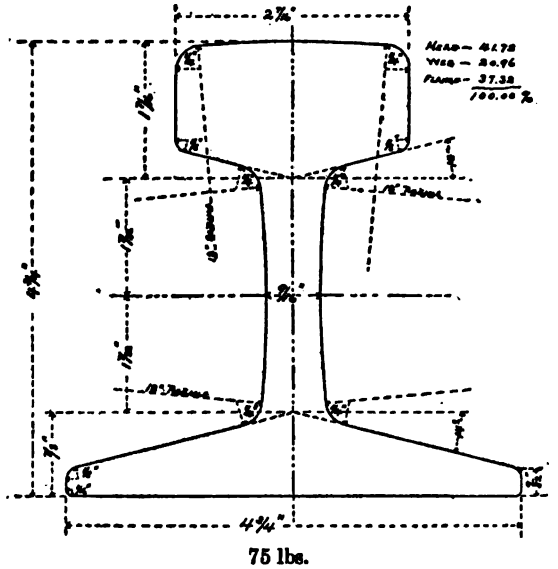
ity and purity. The contrary has proved to be the case. I have ex-

FIG. 3.



70 lbs.

FIG. 4.

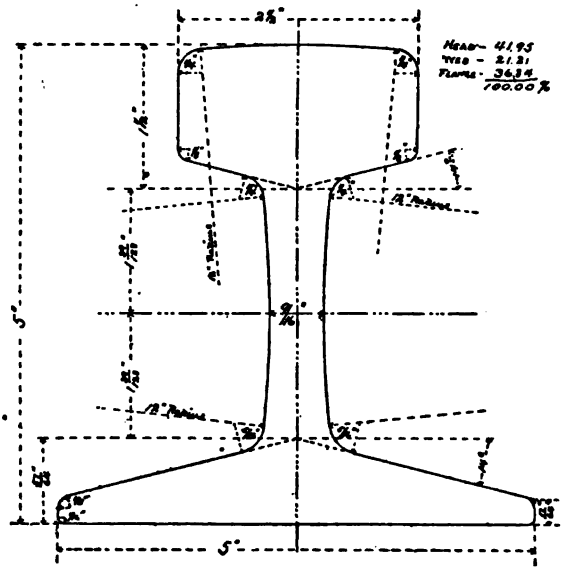


75 lbs.

amined the analyses of numbers of these rails which had been removed

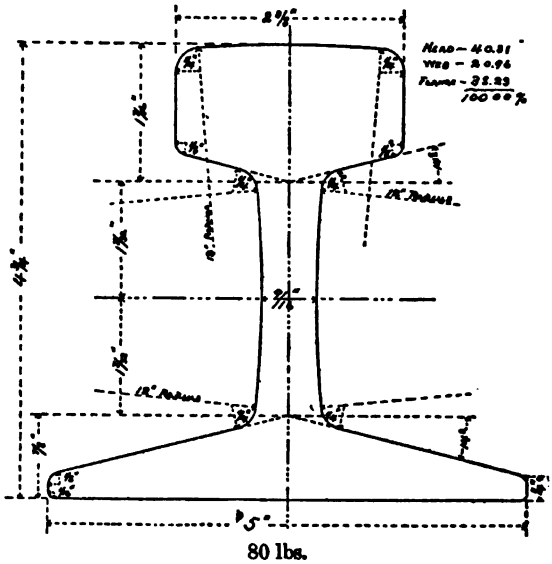
from the track after many years of good wear, and had been selected

FIG. 5.



75 lbs.

FIG. 6.

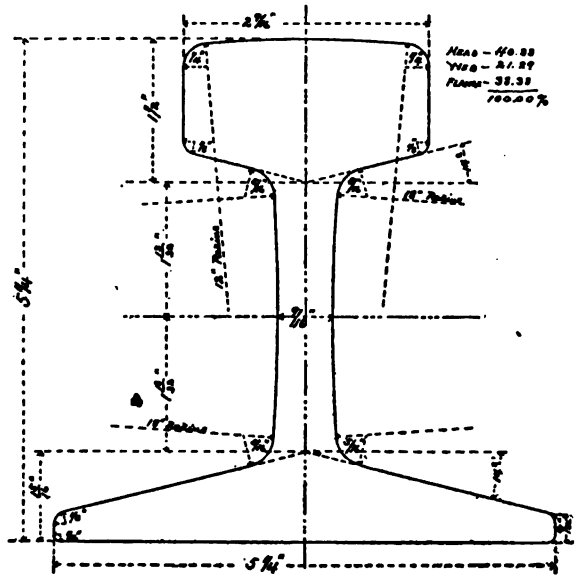


80 lbs.

as representing ideal rails. When we consider that the carbon

varied from 0.24 to 0.70 per cent., the silicon from 0.032 to 0.306 per cent., the phosphorus from 0.077 to 0.156 per cent., the sulphur from 0.050 to 0.181 per cent., and the manganese from 0.312 to 1.046 per cent., we are forced to the conclusion that their chemical composition was not the controlling influence in their good character. I am therefore strengthened in my belief that the sections and physical treatment of the steel during manufacture must account for the results obtained. As I have stated on previous occasions, the early steel-makers did not know so much more than the later ones, as

FIG. 7.



85 lbs.

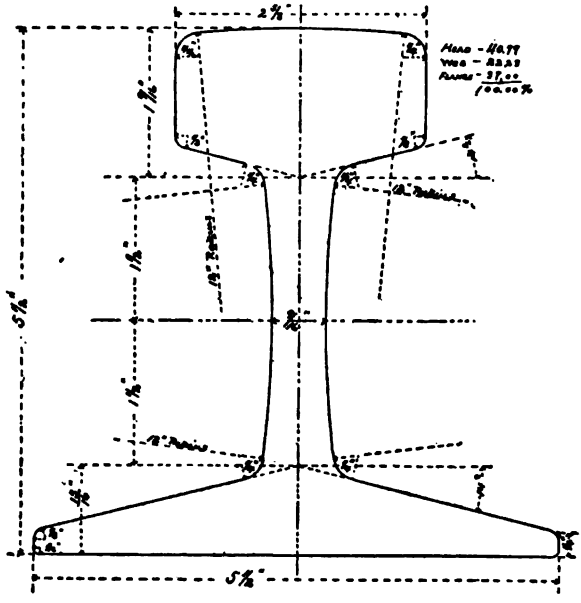
might be inferred from the frequent praise bestowed upon their work. On the contrary, they had much less knowledge, and in consequence were afraid to depart radically from the existing traditions of steel-making.

I do not claim originality in advocating broader-headed rail-sections. The present well-known standard 80-lb. rail of the New York Central and Hudson River Railroad, was designed by our fellow-member, Mr. P. H. Dudley, in April, 1883. I filled an order at the Troy works in 1884 for 1000 tons of this section. The rails were laid through the Fourth Avenue tunnel of the Hudson River

road, extending to the junction with the Harlem road. Rails of other and lighter sections had given unsatisfactory results in this locality. After one year's experience, the company above named adopted the Dudley section as its standard 80-lb. rail, and since then has put down several thousand tons each year. In fact, it is relaying the whole Hudson River division with this section.

As is well-known, Mr. Dudley was the first to use the dynagraph car in determining the condition of road-beds, and has devoted himself to that branch of his profession for many years. His paper on "Railway Resistances," read before the Institute at Washington, in February, 1876 (*Trans.*, iv., 232), was the product of much ex-

FIG. 8.



90 lbs.

perience already obtained in the use of his ingenious invention, and has been followed by extensive and continuous applications of it, with results of the greatest value to our railroad engineers. His observations convinced him that, to meet successfully the tremendously increased wheel-pressure, rails must present a broader head-surface. As a result we have his 80-lb. rail. As will be seen upon examining my 80-lb. section, I have not adhered to the Dudley rail, but have approached more closely to the one adopted for the Michigan

Central Railroad by Chief Engineer J. D. Hawks. In designing all my sections, I have kept in view the controlling circumstances of their manufacture, adopting wide and comparatively shallow heads, strong webs, and flanges with thick edges. I have tried to distribute the metal so as to avoid cooling-strains and give rails which will require the minimum of cold-straightening.

As at present conducted (and mechanical talent has as yet failed to find a better way), cold-straightening is the enemy of a perfect rail. It is generally accepted that the hardness of the steel must be increased to improve its wearing qualities. But this augments the danger of the breaking of cold-straightened rails at the "gag-marks." And if, in seeking to avoid such a consequence, the straightening is performed while the steel is too hot, "lumpy" or "wavy" rails will be the result.

Figs. 1 to 8 inclusive present my designs for 60, 65, 70, 75, 80, 85, and 90-lb. sections. There are two 75-lb. rails (Figs. 4 and 5), the first with a $4\frac{3}{4}$-in., and the other with a 5-in. base. The latter is intended to meet the views of those engineers, who, because they are compelled to use soft wood ties, or for other reasons, require that width of base.

These sections are all designed to be used with fish-plates. When the time comes, as I believe it ought to come, and will, that joints affording a base support are used, we can discuss a very different rail. To meet the conditions which will prevail until then, I respectfully submit these suggested forms.

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*NOTE ON THE INFLUENCE OF COLUMBITE UPON THE TIN-ASSAY.**

BY FRANKLIN R. CARPENTER AND W. P. HEADDEN, RAPID CITY, DAKOTA.

(New York Meeting, February, 1889.)

DISCUSSION.

WM. P. BLAKE, New Haven, Conn. (Communication to the Secretary, June, 1889): It is gratifying to find that Prof. Carpenter in his paper sustains the accuracy of my original determination of

* NOTE BY THE SECRETARY.—Prof. Blake's communication was received too late for insertion in its proper place. See page 633 of the present volume.

the black tabular mineral occurring at the Etta Tin mine as columbite rather than tantalite, as determined by Prof. Schaeffer. The different conclusions may perhaps be explained when we consider that the Etta hill may afford specimens quite different in their composition. My specimens were obtained from a mass of albite and quartz in one of the tunnels or drifts, and except at that point the mineral is rarely seen, and I did not leave much of it behind me. I did, however, find a few small crystals in the greisen at other points, but not enough to cause any anxiety for the purity of the cassiterite or the tin smelted from it, even should the columbite prove to be a noxious impurity. The discussion therefore resolves itself into a matter of interest to science rather than one of practical importance to the metallurgy of tin.

My reasoning that neither columbic or tantalic acid would be reduced so as to contaminate the tin in smelting was *a priori* and not based on actual trials. It is therefore the more interesting and satisfactory that the question is under careful investigation by Prof. Carpenter and Mr. Headden. It is evident that there must be a careful analysis of the mineral to determine the percentage of tin it contains, and that the buttons must be carefully analyzed to determine the nature of the substance or substances which cause the increase in the weight of the buttons when the columbite is added. It will probably be found that the increase is due chiefly to the iron and manganese reduced by the powerful cyanide flux at a high temperature—two conditions not found in the commercial tin smelting.

Dr. Carpenter, while conceding the justness of my statement that the columbite can be separated from the tin stone by hand, says it is usually present with the stream-tin. So far as my observation has extended, it does not occur in the stream-tin from Nigger Hill, which is by far, up to this time, the most important source of stream-tin in the Hills. It is also important to observe that the block-tin smelted from this Nigger Hill stream-tin was remarkably pure, clean, and satisfactory. In some of the stream-tin of other localities there is a notable quantity of columbite and of other ferruginous minerals, perhaps ilmenite, rutile, etc., and wolfram is not unknown. No two valleys in the tin region afford exactly the same product in stream-tin. The same is true of the different veins. The cassiterite varies in color and aggregation. Some veins afford well-crystallized masses, in others the cassiterite is amorphous and massive. At the Cow Boy the ore is reddish and is in a gangue of

quartz, not of granite or greisen. Careful analyses of the tin stones of the different veins are desirable and no doubt will show considerble differences in nature of the impurities present. Berzelius found over 2 per cent. of tantalic acid in a cassiterite from Finbo, Sweden.

END-LINES AND SIDE-LINES IN THE U. S. MINING LAW.

BY R. W. RAYMOND, NEW YORK CITY.

(New York Meeting, February, 1889.)

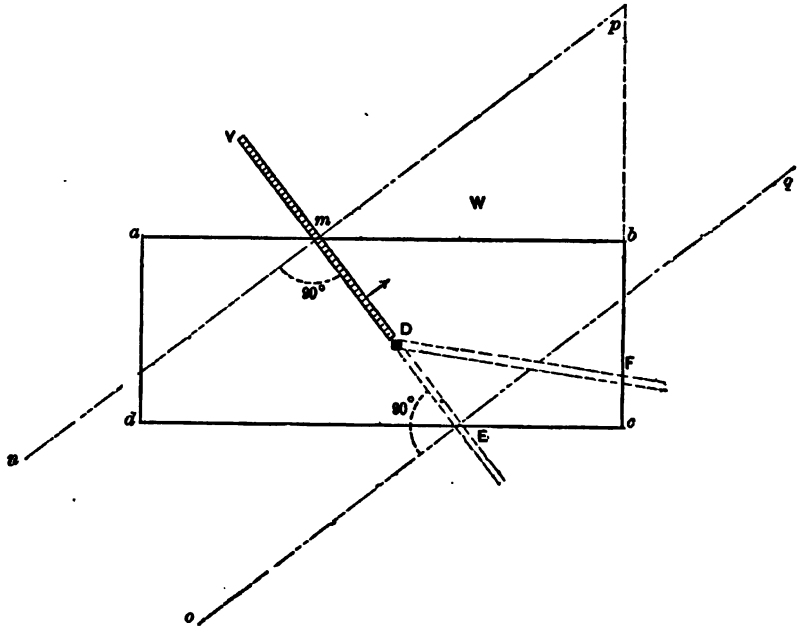
THERE is apparently no end to the doubts, inconsistencies and absurdities in which the courts of our mining States and Territories are involved in their attempts to apply to conditions of ever-increasing complexity the provisions of our anomalous mining law. In previous papers, I have called this statute "the law of the apex," because it based upon that vague, novel and mysterious condition the most important grant of title. But the apex is scarcely a more essential feature than the end-lines and the side-lines; for the rights which it may confer may be limited in various ways by these; so that we have, in many a case, the most ridiculous result that the United States, in return for the money of a citizen, conveys to him with much solemnity, under its broad seal, nobody knows what, bounded nobody can tell how.

A decision recently made at Butte by Judge De Wolfe, of the United States District Court of Montana, in the case of *King et al.* vs. *The Amy and Silversmith Mining Company*, furnishes a fresh illustration of this criticism. The diagram on the following page, constructed without precise regard to courses and distances, simply to assist in the statement of the essential features of the case, will, I hope, make clear the principles involved:

The plaintiff occupies a location (the Nonconsolidated) adjoining on the north the defendant's location, *a b c d* (the Amy), and the disputed mining-ground lies, let us say, in depth under the point *W*, that is, on the dip of the vein, *V D*, but not under the surface-location, *a b c d*. The defendant has the prior title, and claims this ground under its extra-lateral right. It is admitted that the apex does not cross the end-line, *a d*, and does cross the side-line, *a b*. Whether it crosses, at the other end of the location, the end-line, *b c*

(say at F), or the side-line, $d\ c$ (say at E), is disputed, and cannot be clearly shown by tracing beyond the discovery-shaft, D. In other words, it is disputed whether the course of the apex across the claim is $V\,D\,E$ or $V\,D\,F$. The Court declares this question to be unimportant, but afterwards assumes that the apex has the course $V\,D\,E$, though the precise position of E is not determined.

The plaintiff contends that the defendant's extra-lateral right is



Imaginary Diagram, to illustrate the Amy and Silversmith Case.

limited by a vertical plane through $a\ b$, which, being crossed by the apex, becomes the end-line of the Amy lode-location. The defendant contends that a vertical plane through m, parallel to $a\ d$ and $b\ c$, is one legal boundary of its extra-lateral right, and (we must suppose) that the other boundary is a plane, either through $b\ c$ or parallel to $b\ c$ through E, according to the facts, when ascertained. The Court adopts neither view, but holds that vertical planes through $n\ p$ and $o\ q$, at right angles to $V\,D\,E$, are the proper boundaries. This view gives the disputed ground to the defendant, though not on the defendant's theory. What else it accomplishes, we will presently consider.

The following extract from the decision covers the whole of its

argument on the subject. After quoting Section 2322 of the Revised Statutes, the Court says:

"The statute, it will be perceived, evidently contemplates that locations upon mineral veins or ledges will be made along and in the direction of the strike or course of the vein, and only provides for following the vein in its dip or downward course where it passes underneath the side-lines of the claim located. It does not mention veins or ledges which cross the surface-ground at right or any lesser angle and defines in express terms the rights of the locator on veins thus crossing the surface-location and in their course or strike departing through their side-lines; but, as seen, it does provide that the locator shall have the 'exclusive right of possession of all such veins or lodes throughout their entire depth, the top or apex of which lies inside the surface-lines of the location extended downward vertically.'

"Now the apex of the vein in question does lie within the surface-lines of the Amy claim, as located from the point at which it enters the surface-location 184 feet east of the west end-line of the Amy claim to some point on the south boundary-line of the same claim (said point, as stated, not being clearly established by the evidence), where it leaves the Amy location and enters the Frank Moulton lode. If the vein is of uniform strike from the Amy discovery-shaft to the point at which it leaves that location and enters the Frank Moulton claim that it is shown by the evidence to be from the discovery-shaft to the point at which it crosses the north side-line of the Amy and enters the Nonconsolidated claim, the length of the vein and of the apex in the Amy claim would, in round numbers, be 800 feet.

"The vein by the concurrent testimony of all the witnesses, has a pitch to the north in its dip or downward course.

"What, then, is the portion of this vein which defendants have the right to follow in the dip, it being conceded in argument, and the law itself being plain on the point, that the defendant has no right to follow the vein on its strike or course after it departs or crosses the side-lines of the location on the surface? As the law gives the locator the apex within the surface-location and the right to follow the vein on its dip or incline into the earth, is the locator not entitled to the same length along the vein on its dip that he has to the apex on the surface? And as the dip of a vein is supposed to be at right-angles with its strike, lines drawn through the vein at right-angles with its strike or course where the vein departs from the surface side-lines of the claim, and these lines extended laterally in the direction of the dip or pitch of the vein beneath the surface, and vertical lines drawn downward through these lines and intersecting the vein beneath the surface, define, in my view, the dip of the vein, and the portion thereof to which the defendant in the present case is entitled.

"This right to follow the vein in its dip is of course limited by the provision of the section referred to as not permitting a locator to pass beyond planes drawn vercally downward through the end-lines of the location and thence extended laterally so as to intersect the exterior parts of the vein or lode.

"It is further believed that neither of the cases referred to by the counsel and decided by the Supreme Court of the United States, wherein it is strenuously urged that a contrary doctrine has been held, asserts any principle or construction of the mining law contrary to that herein maintained; but time is not at present granted to examine or reconcile, or attempt to reconcile, those decisions with the views here expressed."

I cannot but presume that the cases alluded to in the last paragraph were the *Flagstaff* and *Elgin* cases, since in these the opinions of the Supreme Court as to end-lines and side-lines have been chiefly, if not exclusively, made known. The *Elgin* decision (delivered by Justice Field, April 26, 1886, in *The Iron Silver Mining Company* vs. *The Elgin Mining and Smelting Company*) particularly deserves to be recalled in this connection; and I would refer for its clear illustration to the diagram of the Stone claim, in dispute in that case, given in my paper on "Lode-Locations" (*Trans.*, xv., 292).

The Iron Silver Mining Company, owning by prior title the Stone claim, asserted an extra-lateral right to the Gilt-Edge, on the dip of that lode. The other side denied that any such right could exist under the circumstances:

1. Because the Stone claim had not two parallel end-lines, as the law requires.

2. Because, if its two nominal end-lines could be accepted as valid (*i.e.*, if *C* were assumed to be on the so-called "south end-line"), vertical planes drawn through them would not include the Gilt-Edge claim.

Chief Justice Waite and Justice Bradley, in a dissenting opinion, observed:

"The end-lines of a mining location are to be projected parallel to each other and cross-wise of the general course of the vein within the surface-limits of the location; and whenever the top or apex of the vein is found within the surface-lines extended vertically downwards, the vein may be followed outside of the vertical side-lines. The end-lines are not necessarily those which are marked on the map as such, but they may be projected at the extreme points where the apex leaves the location as marked on the surface."

As I remarked in the paper cited, this opinion apparently implies the propriety of rectifying all end-lines. In that respect the opinion of Justice Field, representing the majority of the Court, as to the controlling force of end-lines once established, in accordance with the statute, is to be preferred. The following passage from that opinion is admirably clear and forcible:

"With all the care possible, the end-lines marked on the surface will often vary greatly from a right-angle to the true course of the vein. But whatever inconvenience or hardship may thus happen, it is better that the boundary-planes should be definitely determined by the lines of the surface-location, than that they should be subject to perpetual readjustment according to subterranean developments made by mine-workings."

On this basis, the Court says it will not permit the actual end-lines

(*i.e.*, the two crossing the apex) to be altered; and because they are not parallel, it denies to the Stone claim all extra-lateral rights.

It thus appears that, concerning valid end-lines for lode-claims, the essential requirements of the statute, as expounded by the Supreme Court, are two: 1. According to the *Flagstaff* decision, they must be "cross-wise of the general course of the vein on the surface" (not necessarily of its true course). 2. According to the *Elgin* decision, they must be parallel. Both these conditions are held essential to the acquirement by the locator of an extra-lateral lode-right, between vertical planes drawn through his surveyed end-lines. But the effect of the non-fulfilment of the first condition is curiously different, according to the *Elgin* decision, from the effect of a non-fulfilment of the second.

In the *Flagstaff* case, the end-lines were parallel, but they were not "cross-wise of the general course of the vein on the surface;" or, to use language more generally applicable, and conveying what we conceive to be the exact meaning of the Court (since most veins do not literally reach the surface, and the apex is "the edge nearest the surface"), the apex did not pass under these end-lines, but passed under two parallel side-lines instead. In the *Flagstaff* case, the Court declared these side-lines to be the true end-lines, and gave the locator the extra-lateral right bounded by vertical planes drawn through them. His error (if it was an error; there is some reason in history for suspecting it to have been intentional) was punished by giving him a lode-claim as complete in all respects, except that of length, as if the location had been correctly made.

In the *Elgin* case, on the other hand, the end-lines (that is, the located north line, and the line, not located as an end-line, at the other extremity of the claim, under which the apex actually passed) were not parallel. Here the far severer penalty of a total loss of all extra-lateral lode-rights was inflicted. Yet the defect in the location had resulted from an attempt to carry out the principles of the *Flagstaff* decision, and make the location follow the surface-course of the vein, which, by the erosion of a ravine across it, had been made to appear bent like a horse-shoe, though the true course was quite uniform. The vein lay like a slice of ham in an inclined sandwich of country-rock; the ravine had bitten out a semi-circle; the *Flagstaff* decision required the location to follow, not the course of the original edge of the ham, but the edge of the ham as left after the bite; and having got, by this circuitous route, around to the other side of the bite from where he began, the locator tried to meet,

as best he could, the second condition, namely, that of parallelism in the end-lines. In his attempt to do this, he missed, by a few feet, the first condition. The apex had been supposed to cross the "south end-line," but, in fact, passed out of the claim at a very near point on the next side of the irregular polygon of the location. If it had crossed the end-line, the location would have been perfect, with two parallel end-lines, both "cross-wise" of the apex. But these lines, continued northeast, would not enclose any part of the Gilt-Edge claim, which was the ground in dispute; and hence the Court did not consider as seriously as it might have done the propriety of condoning the small error of the survey, and of declaring the south end-line valid. This would have been in the spirit of a passage in the *Flagstaff* decision which speaks of "slight deviations of the outcropping lode from the location" as "probably" not affecting the locator's rights—although that passage does not literally apply to this case. But as the Stone locator was bound to be beaten "in either view" of his south end-line, the Court did not pause to do more than recognize, in passing, the possibility of the view which would admit the validity of that line. The line did not comply with the law, strictly construed; and a liberal construction on that point would have done nobody any good. Yet, as I have formerly said, I think the Court might have adopted that construction with propriety. It would not have affected the result to the litigants; but the precedent, being founded on the extremely peculiar circumstances of this particular case, would not have bothered the Court subsequently, as its actual decision is likely to do.

All the judges agreeing that the south end-line was not valid, the minority said that both end-lines ought to be rectified by substituting lines drawn "cross-wise of the general course of the vein within the surface-limits of the location." If this curious phrase means *at right-angles with the true course of the vein*, the rectified end-lines would be drawn parallel to the dip. But "cross-wise" does not necessarily mean at right-angles. The majority decision explicitly so declares; and the adoption in the dissenting opinion, immediately thereafter, of the ambiguous term *cross-wise*, instead of the precise term *at right-angles*, seems to be significant. Hence, the north end-line is already "cross-wise" of the apex and of the vein, on any hypothesis as to the course of either, and, under the language of the dissenting opinion, it would not be really necessary to rectify that line at all, but only to draw a new south end-line parallel to it at *C*. This is what I suggested ("Lode-Locations," *Trans.*, xv., 303),

as a fair application of the statute to such cases, in the following words:

"When one end-line crosses the apex of the located lode, and the other does not, but the apex departs from the claim across a side-line, not parallel with the valid end-line, and the deviation is too great to be overlooked as unimportant, then the boundaries of the extra-lateral right shall be a vertical plane drawn through the valid end-line, and a vertical plane, parallel thereto, drawn through the point where the apex of the lode passes under and across the side-line of the claim."

But the majority decision, starting from the invalidity of the south end-line, and considering the line crossed by the apex to be the real end-line of the location, says the two end-lines "are not and cannot be made parallel," and on this remarkable statement bases its startling conclusion that the location and patent carry no rights whatever, either to the mining ground in dispute or to any other ground outside of the surface-location. I have formerly pointed out the unfortunate effects of this decision, if it should be followed as a general precedent. I wish now to call attention to the peculiar statement which constitutes its foundation. That the two lines to which it refers are not parallel, is clear enough, even to a layman; but what does the Court mean by saying they "cannot be made so?" Does it mean that, to a location of this shape, following the apex around a curve of some 135 degrees, it is impossible to draw two parallel end-lines, cross-wise of the course of the apex? That is true only if the lines must cross the apex *at the same angle*. Otherwise, it is not true at all. The Court has declared that "cross-wise" does not necessarily mean at right-angles. Why should it then mean any particular angle for both lines, or for either? Or, does the Court say that these lines cannot be made parallel, because the law will not permit any change to be made in their direction, after they have once been established? That would be more comprehensible; but it is not indicated by the context. The declaration has the air of the statement of a physical fact, rather than a legal principle. It occurs in the presentation of the facts of the particular case, after an exhaustive discussion of the principles upon which they are to be adjudicated.

On the whole, we seem to be driven to the hypothesis that the declaration that these lines not only "are not," but "cannot be made, parallel," has only a rhetorical significance. It is merely an emphatic expletive, as who should say, "Them lines ain't parallel, and you can't make nothin' else out of it!" or "There ain't no end at

all, and there's an end on't!" Such asseverations consort better, it is true, with the vigor of the street than with the dignity of the bench. And perhaps there is a quiet implied rebuke in the dissenting opinion, which says in substance, "They are not parallel, but they can be, and ought to be made so."

Judge De Wolfe's decision appears to apply the principle of the dissenting opinion in the *Elgin* case. That the majority opinion in that peculiar case may be much modified by the Supreme Court itself in application to other conditions, is certainly not impossible, perhaps not unlikely; but that such modifications will ever go so far as to permit the "projection" of new boundary-planes, instead of planes through such surveyed location-lines as actually cross the apex, seems to us improbable in a high degree. Still more improbable would be the alteration of a boundary once fixed by an original, valid end-line. In other words, if, of a rectangular claim, one surveyed end-line duly crosses the apex, and the other does not, but a side-line does, the Supreme Court may hold, as it has held in the *Elgin* case, that there is no extra-lateral right; or it may consent to a boundary-plane through the intersection of the apex and the side-line, drawn parallel to the valid end-line; but we do not think it will consent in such a case to two new boundary-planes.

Was the case decided by Judge De Wolfe such a case? As I have observed, the Court did not know, but said that that made no difference; that whether the lode crossed two side-lines, or one side-line and one end-line, there must be two new boundary-planes, placed at right-angles across the apex.

But it may make a great difference which of these two statements represents the facts, although, on either hypothesis, the Supreme Court may easily find reason to reverse the Montana ruling. For if the apex crosses two parallel side-lines, then these are (by the *Flagstaff* decision) true end-lines, and the boundary-planes must be drawn through them. On that hypothesis, Judge De Wolfe would have had before him, apparently, the precise *Flagstaff* case, and decided it on a principle opposed to the *Flagstaff* decision. On the other hypothesis—of an apex crossing a side-line and an end-line, not parallel,—his decision runs against the *Elgin* majority opinion, the principles of which will not be easily so far modified as to include this variation.

Apart from the question of precedents, some further remarks are suggested by this *Silversmith* case.

1. The court is here undertaking to determine the proper vertical

boundary-plane at one end of the claim, when the strike and dip of the lode at the other end are not known. The location of *E* on the line *d c* being undetermined, the general course of the apex across the claim is not determined. Consequently, the Court must mean (as its words do indeed literally assert) that the boundary-plane through *m* should be at right-angles with the strike *as shown at that point*. But this plane is of course to be a permanent boundary; and since, as the Court explicitly declares, the locator should have the same length along the lode in depth as along the apex, it follows that the other boundary-plane, hereafter to be determined (say at *E*), must be made parallel to the one now determined through *m*.

2. But suppose that the apex should cross at *F*, on *b c*, not on *d c*. In that case, this decision would either abolish a perfectly valid, surveyed end-line, established by the patent, or give two boundary-planes not parallel.

3. Either the plaintiff's contention, denying all extra-lateral rights north of *a b*, or the defendant's contention, that the boundary through *m* should be parallel with the surveyed end-lines, *b c* and *a d*, would avoid the difficulty above suggested; for in either case, the decision would be independent of any determination of the local or the general course of the apex, beyond the one fact that at a known point *m* it crosses a side-line. That being admitted, the surveyed side-lines or the surveyed end-lines would determine, on the respective theories, the rights of the parties.

4. The last paragraph but one in this decision, if I understand its meaning, declares that while the surveyed end-lines cannot serve to fix, for a lode-claim in which they are not crossed by the apex, the boundaries of the extra-lateral right, they do, even in such a case, fix the boundaries beyond which that extra-lateral right, however determined, *can never go*. In other words, as I understand it, the Court gives to the defendant the boundary-plane *n p* through *m*, but adds, that the defendant's right to follow the lode in depth is stopped absolutely by the plane through *b c*. This gives the defendant, outside of its surface-location, only the ground under the triangle *b m p*—a curious result of the professed attempt to secure to him the same length of lode at each level in depth as along the apex.

It is not always either fair or useful to blames judges for such results. The statute itself is the source of hopeless confusion; and the most ingenious attempts to construe it in accordance with the equities of particular cases only aggravate the difficulty, by furnishing precedents which, in other cases, become absurd. I may appro-

priately repeat, at this point, the words in which I described, more than two years ago ("Lode-Locations," *Trans.*, xv., 304), the situation, since that time grown rather worse than better:

"The law of the apex and its extra-lateral right are prolific of difficulties, doubts, contradictions and absurdities. Every judge that administers it utters a fervent hope for its amendment. But the only effective amendment would be the introduction of that system under which the mine-owners of the east and the placer-owners of the west get along in complete security and peace. Let the government cease to grant a thing which it cannot define. Let the citizen know, without the aid of geologists, lawyers, juries and courts, what he has got when he has located and purchased a claim. Make the lode laws resemble the placer laws, and the coal-land laws and the homestead and pre-emption and other agricultural land laws, and the mining land laws of all other civilized countries. Banish that illusive thing, the apex, to the limbo of vanity, where it belongs. Put an end to the supernatural extra-lateral properties of end-lines, and their infinite projection into space. Let side-lines remain in that situation to which Providence and the surveyor have called them, without danger of being suddenly exalted into end-lines, possessed of transcendental potencies. In short, let us have a mining law, and not a chaos of confusion, ameliorated at long intervals by the well-meaning interference of the Supreme Court, which sets to rights a little corner of the hubbub, and necessarily raises as much dust as it lays. We are slowly getting authoritative decisions of one point after another; but the best thing about them is, that they show us with increasing clearness how bad the law is."

Or, more briefly, the "square location" is the "square thing!"

But the direction of the lines of a location, as shown by the surface-survey, in their relation to the course of an apex, is not the only element of doubt concerning their effect. Having busied myself for many years in trying to foresee the possible complications under this law, I must confess myself to have been taken by surprise after all, by a novel and interesting decision, given orally by Judge Hallett, November 30, 1888, in the United States Circuit Court of Colorado, in the case of *Nolan et al.* vs. *The Colorado Central Mining Company.* I have only a newspaper report, which appears, however, to have been stenographically made, and perhaps, also, revised by the judge. At all events, whether it be perfectly accurate or not as to names and figures, its statement of the legal points considered is entirely clear and self-consistent. It seems to have been the fortune of the District Court of Colorado to encounter a larger number of the subtle and difficult questions involved in our mining law, and to offer more ingenious solutions of them, than any other single tribunal. This is no doubt largely due to the peculiar conditions obtaining in that State, which presents locations of all sizes, shapes and ages; local mining customs antedating all the statutes; territorial laws supple-

menting the Federal law; ore-deposits combining great irregularity with great value; numerous costly, important and desperately contested litigations over mining titles and boundaries; a State law giving the defeated party in a suit involving the title to real estate the right to demand a second trial; and litigants, lawyers and juries who are skilled in the business and fond of it.

On the other hand, I am inclined to believe that Judge Hallett himself has peculiarities which correspond with these peculiar conditions, and which have contributed to the abundance of fine distinctions and curious definitions propounded in this Court. In the case just mentioned, he discovered a principle which had not occurred to the counsel on either side, and, for lack of evidence on the point raised by the Court, dismissed the application of the plaintiffs (apparently for an injunction) "with leave to renew." This illustrates both the exceptional acuteness of the Court, and the inexhaustible resources of the law.

The case having thus been decided on a single point (and that point not known to be in it), it can be easily stripped of all collateral questions, and stated, as to its controlling issue, thus:

X, the mining ground in dispute, is affected by the following proceedings:

a. A located, prior to 1879, a claim including X.

b. B made in 1879 another location, the lines of which include X.

c. B applied for a patent in 1882, explicitly excepting X from his application, because it was covered by an earlier location.

d. Subsequently to B's application, A's location was changed so that A's claim to X was formally abandoned.

e. Apparently B did not know this; or, perhaps, he knew it, but did not care to delay proceedings by modifying his application. At all events, the patent was issued to B, according to his first request, for a certain tract (presumably a parallelogram) *excepting and reserving* X, which was described as belonging to the earlier locator (though he had already abandoned it) and was neither asked for by B, nor granted to him. The case is evidently stronger than if X were any other piece of ground, concerning which B had had no communication with the government; for the record exhibits, first, the solemn declaration of B to the government: "Don't misunderstand me, *I do not want X*," and the government's solemn reply, to prevent all misunderstanding: "We hereby declare that we do not give you X."

f. B accepts his patent, and pays, at so much per acre, for all the

land within its survey, excepting X. (This practice of issuing such ridiculous grants, which first describe more land than is conveyed, and then subtract what is not conveyed, is one of the beautiful results of the original features of the law, combined with the still more original constructions and regulations of the General Land Office.)

g. C, D and E subsequently locate parts or the whole of X, at different times, C coming first.

h. The later locations play no part in the decision of the principal issue. If C's location be regular and valid, they can scarcely have any force against it. At all events, the Court brushed them aside for the time being, as not needing consideration, in view of the relative *status* of the two parties, as will appear below.

i. B (that is, the Colorado Central Consolidated Mining Company, which was either the original B, or entitled, as his grantee, to all his rights), in working upon the patented claim (described in paragraph *f* above), followed the lode upon its course into the ground covered by C's location; and C began an action in equity, and asked for an injunction restricting B from working in that ground.

All cases of this kind being in the nature of actions of "trespass" or "ejectment," the plaintiff must recover on the strength of his own title; not the weakness of his adversary's. This rule is so obviously just that one would scarcely expect it to be overlooked in mining litigation. "The defendant is on *my* property; make him get off!" is the essence of the complaint; and the first question of Justice must be, "Is it *your* property?" To prove the defendant a thief, a tramp, a trespasser on somebody else, is not in the remotest degree pertinent. Yet one or two important cases have got as far as the United States Supreme Court, to be knocked on the head by this simple proposition; and the notion of many mining managers, and of their lawyers, too, seems to be that opponents cannot become dangerous if they have bad titles. That depends. They can be much more dangerous in mining cases than in any others, for the reason that if the title of the party they trespass upon be not perfect, and the real ownership of the ground be still in the United States, no one having as yet a valid title to it, they are then, under the statute, not trespassers at all, but licensed explorers.

In the present case, therefore, C had to prove title, and presented, for this purpose, proof of his location. It may be inferred that B, in attacking this title, pleaded his own patent for what it might be worth, and brought forward also D and E, involving, no doubt, a multitude of small points, both of statutory and of "Department"

law. Of course, since any doubt as to C's title would defeat his motion, one such doubt was as good as many; and the Court suggested one, which thus took the place of all the rest.

I quote the report of Judge Hallett's words on this point:

"The first question presented, therefore, is whether defendant as the owner of the Colorado Central Extension lode can have any interest whatever in this territory. That is a very peculiar question, because the exact circumstances which exist in this case do not often arise. Usually, when territory is excluded from one location, because it is included within another and prior location, a patent has been issued for the other and prior location, thus granting the territory absolutely to the earlier location, and that, under the statute, covering not only the principal vein in the earlier and prior location, but all others having their outcrop within the territory of that location, takes everything within the territory.

"It seems, however, that the withdrawal of the Nolan-Gilmer location from the territory which, at the time of the application for patent for the Colorado Extension lode, was supposed to be within that location, may leave the complainants some right therein which otherwise they would not have had against the Nolan and Gilmer claim. For the grant in this patent for the Colorado Extension lode, while it excludes from its terms the territory covered by the several locations in conflict with this one and the Nolan-Gilmer location, conveys in express terms 1000 linear feet of the Colorado Central Extension lode. That was the length of the claim, 1000 feet. And the exclusion of the territory of other locations, that is, the surface-ground of other locations, from the surface-ground of this one, may not affect the grant of the lode for the length defined, if it had not been previously granted to other claimants of the conflicting territory. It is not true that the surface-ground granted by a patent is always coincident or coextensive with the vein which is conveyed by the patent. As illustrating that, reference may be made to Section 14 of the Act of 1872 concerning mining lands, the familiar provision that,

"'Where two or three veins intersect or cross each other, priority of title shall govern, and such prior location shall be entitled to all ore or mineral contained within the space of intersection, but the subsequent location shall have the right of way through the space of intersection for the purpose of the convenient working of the mine.'

"This has been held to mean that, in such case, the junior location may have the vein within the territory of the senior location up to the very point of intersection between the veins. In other words, it is not the space of intersection of the location, but the space of intersection of the veins, to which the statute refers; and very clearly, under that section, where there are cross-veins, the junior patent conveys the vein within the territory of the senior patent, except at the very space of intersection of the veins. That is a case in which the patent conveys the vein to a greater extent than it conveys surface-ground, and I apprehend that it may be practicable, under the law, and by patents which are issued pursuant to the law, to convey a lode of the length of 1500 feet with only surface-ground at the very point of discovery, if any one should choose to make a location of that character; and to convey it in a way that other and subsequent locators upon the territory in which the vein outcrops might not get any interest in it.

"Upon the question that arises in this case, it seems to me that, under this patent, if the vein of the Colorado Extension lode on its strike extends into and through the territory in dispute, within the lines of that location, and which was within the

Nolan-Gilmer location, at the time the patent was applied for, and at the time it was issued, the defendant now owning that title may hold it, and that it would not be subject to relocation by any one coming subsequently to the entry or the issuance of the patent.

"This, I presume, cannot be true as to the other lodes, if any, existing within the same territory, because the grants of those lodes attending upon the grant of the principal lode, as I understand the statute, pass only with the surface-ground; therefore, the patent is not the same as if it conveyed that territory, because the patentees do not gain the other lodes in that territory, that is, those other than the Colorado Extension lode upon which the location was made.

"The materiality of this question arises upon the point whether the place in which defendants are now working, and from which the bill seeks to enjoin them, is upon the Colorado Extension lode or not; and upon that question the bill and all the papers in the case are absolutely silent. It was not supposed by counsel presenting the case, I presume, that any such question could arise in this controversy. In making the locations, and in the bill upon which complainants rely, it is assumed that the locators get whatever there may be within this territory, and upon proof that the ore-body on which the defendant is working outcrops within the space of intersection, between the Nolan-Gilmer and the Colorado Extension lode, that the owners of the new locations would be entitled to take it.

"As I think the fact is otherwise, and that if the ore-body is upon the Colorado Extension vein, that is to say, that vein upon which the location was made, the defendant holding title under this patent may claim it, I must deny the application for an injunction."

Upon this decision I offer the following remarks (assuming the word "complainants," in the first sentence of the second paragraph, to be, as the context seems to prove beyond doubt, a reporter's error for "defendants"):

1. The judge pronounces this case peculiar, because the original location of A was never patented. A patent, he says, would have granted to A "everything within the territory." The justice of this distinction is not apparent. Every right granted by a patent is explicitly given by Section 2322 of the Revised Statutes to the *location*. The words are: "The locators of all mining locations . . . . shall have the exclusive right . . . . of all veins, lodes, and ledges," etc. And, strangely enough, the statute says nothing at all about any such right as given by the patent. It is absolutely certain that if the location does not possess it, the patent cannot give it. I am not aware of any difference produced by a patent, except that it relieves the owner from complying with certain conditions otherwise necessary for the maintenance of these rights. If A had had a patent, and had afterwards surrendered it to the government, he would have possessed before the surrender, and the government would have recovered by the surrender, no more and no less than was involved in his creation and abandonment of a location.

2. The proposition that the provision of the statute concerning "the space of intersection" common to two veins, means that a junior location "may have the vein within the territory of the senior location up to the very point of intersection between the veins," is true or untrue, according to circumstances. To this case it presents no parallel, and we are forced to conclude that the Court has totally misunderstood its meaning as necessarily fixed by the Act of 1872. A plain statement will show the grounds of this conclusion.

Under the Act of 1866, the surface-location being merely an easement for the working of the lode claimed, locations might be, and were, laid one upon another to any extent. I will not pause to discuss the question whether this was really proper, even under that Act. It was certainly done on the theory that the lode claimed under each location was in fact a separate parallel or cross-lode. Moreover, on the plea that "the lode was the main thing," it was argued that a locator might follow and take the number of feet on its course which he had claimed, whether the lines of his location covered so much or not. But this notion was summarily squelched, in the Flagstaff case, by the United States Supreme Court, which declared that neither the Act of 1866, nor that of 1872, would permit a locator to claim his vein *on its strike* beyond the vertical boundary-planes fixed by his survey or (as we may perhaps infer) properly deducible therefrom.

So long as the surface-surveys of two lodes might cover certain ground in common, the rights of the parties under the Act of 1866 were theoretically plain. On the surface, the elder locator had the prior claim to such of the common area as he needed for buildings, machinery, dumps, etc. Underground, each locator was entitled to his own vein. The only difficulty arose when, by the intersection or union of two veins, the identification of each became difficult. I can recall two cases of this kind, even prior to 1866 and under the old miners' customs. Experts just reveled in them. In one instance, both sides claimed a vein produced by the union of two; and the learned mineralogical grounds on which each party recognized its own vein in the product, and proved the other to be "only a spur," were bewildering enough. In another case, two veins crossed nearly at right-angles, and a valuable bonanza, which characterized the crossing, was claimed by both locators, each insisting that his vein went through the other. Geology, not priority of title, was the test applied.

To do away with this nonsense, the provision was made that, in all

such cases, the prior locator should be entitled to what would otherwise be claimed for both.

Now this provision has, under the Act of 1872, no meaning at all in cases where one lode joins or intersects another *on the strike*. From the moment of location, a locator is entitled to everything under his surface-ground, except lodes which enter that space *on the dip* (in their "course downward"), and have their apexes outside. The case cited as an illustration by Judge Hallett cannot occur.

The source of his error seems to be that he finds this provision in the law, thinks it means something, doesn't perceive what else it can mean, and therefore puts this signification upon it. But it does mean something else, as clearly as it cannot mean what he imagines. It is intended to cover, and does cover, the union or intersection of veins *on the dip*. To take a simple instance, suppose two lodes with parallel outcrops, but differing degrees of dip; No. 2, the later location, being in the footwall of No. 1, and having the smaller (*i.e.*, the flatter) dip. Suppose, further, the two locations to be so laid that each contains from end to end the apex of its own lode, and no part of the apex of the other. It follows, the dips being maintained without change, that at some depth, No. 1, the steeper lode, will be crossed or joined by No. 2. Obviously, this cannot occur under the surface-location of No. 2, it may occur under location No. 1, or under some ground, claimed or unclaimed, beyond. No matter where the junction takes place, the locations of No. 1 and No. 2 are within the extra-lateral rights appurtenant to their apexes, and are thus protected, as to the ownership of the lodes, against all the world —*but not against each other*. To define the relation between No. 1 and No. 2, the law provides that, as to the ore common to both —that is, ore which can be traced to either apex—No. 1 shall own it, and No. 2 shall have the right of way through it.

Judge Hallett's profound declaration that "it is not the space of intersection of the locations, but the space of intersection of the veins, to which the statute refers," is thus seen to be so true as to be entirely inapplicable to his argument. For there is no such thing contemplated or possible under the law of 1872 (and 1889) as a space of intersection between two locations; and there is no right of way conferred upon a junior locator by which, under any conceivable circumstances, he can follow a vein *on its strike* beyond the boundary-planes of his own location.

To put this beyond doubt, we take a more complicated case. Let lodes No. 1 and No. 2 cross one another on the strike as well as the

dip; No. 2 location lying across the north end of No. 1, like the top of a T, and No. 2 dipping south under the location of No. 1. In this case, No. 2 has an extra-lateral right under No. 1; but No. 1 has no right at all under No. 2. The "space of intersection" under location No. 1 belongs to No. 1, and the right of way to No. 2; but under No. 2 the provision has no application at all, every possible right of location No. 1 being cut off by the boundary-plane drawn through the side-line common to both locations.

The Act of 1872, it must be remembered, was framed to meet certain difficulties encountered in the administration of the Act of 1866. One of these was the unintended injury done to the rights of a prior locator, by putting them at the mercy of fine geological distinctions, quite unknown at the time of original location. The fundamental requisite of all titles is certainty and definiteness. The United States, as the owner of the public domain, was not originally bound to give it away or sell it to any one. But having adopted a liberal policy, under which it offers lands under certain conditions free to actual settlers or soldiers, and under certain other conditions at very low rates to purchasers, it is both justified in insisting that these conditions shall be fulfilled, and bound, on the other hand, to give a perfect and definite title to its grantees. No amount of generosity can relieve it from the obligations of justice. A purchaser of the United States title to a mining location, having traced the course of an apex within it, cannot possibly know in advance whether this apex belongs, geologically speaking, to a main lode or to a spur. Some other lode, subsequently discovered, and having an apex in some other location, may be, or may be declared by experts to be, the main lode. It is not only a hardship but also a wrong, that the prior title should be made to depend upon the later development. To prevent this wrong, the Act of 1872 (incorporated, as to this provision, in the Revised Statutes) applied two means: First, as to intersecting *apexes* (that is, as to all lodes intersecting on the strike), it laid down the principle of controlling end-lines; secondly, as to lodes intersecting on the dip, from apexes in different locations, it declared that prior title, not expert opinions, should govern the ownership of the space of intersection. That intersections *on the dip* are meant by Section 2336 of the Revised Statutes (identical with Section 14 of the Act of 1872), is clearly proved by its language concerning the union of veins. Two or more veins unite, forming one vein. If this union is on the strike, does the law give to the prior locator the lode *beyond?* Not at all; the end-line provision settles

that case completely. Whether one vein or many, uniting or crossing, all lodes having their apexes between the end-lines of a location belong to that locator between the end-planes. Hence the statute very properly says that when two or more veins unite, the prior location shall take the vein "*below*" the point of union. That provision is necessary, because two lodes, more or less parallel, having their apexes in different claims, may unite in depth, and thus require a decision, on some principle or other, as to the relation of the extra-lateral rights from the two apexes. Veins uniting present the same two classes as veins intersecting. The first class is covered by the end-lines; the second, by priority of title. Judge Hallett has made the mistake of supposing that the first class of cases, for which the law of 1872 provided an entirely different remedy, is the class contemplated by the passage he quotes.

3. How much weight this mistake may have had in determining the decision, I cannot undertake to say. That it is a mistake, does not necessarily prove that the main proposition which it illustrates is incorrect. Let us therefore examine that proposition more closely. It is, that the withdrawal of A's location from the territory X, included within the survey, but excluded by the terms, of B's application and patent, leaves B some right therein, which he would not have had against A's claim, because B's patent expressly conveys to him 1000 linear feet of a certain lode, and hence, if this 1000 feet of the lode, followed on its strike, would carry him into the ground X, but for the claim of A, he may follow his lode into that ground, and hold it against later claimants, to the extent of that 1000 feet. But this, the court presumes, "cannot be true as to the other lodes, if any, existing within the same territory, because the grants of those lodes attending upon the grant of the principal lode, . . . pass only with the surface-ground; therefore the patent is not the same as if it conveyed that territory, because the patentees do not gain the other lodes in that territory, that is, those other than the Colorado Extension lode, upon which the location [B's] was made."

We encounter here a misconception which has muddled many a local judge—the notion, namely, that a patent grants a principal lode, with an attendant grant of other lodes; and that on the principal lode, it grants a certain number of feet. There seems to be a vague impression that the Act of 1866 has somehow infected the Act of 1872, and that the solemn absurdity of the form of patent in which the Land Office still persists, has some sort of legal force; that when it fixes the length of a location by words and diagrams, accord-

ing to an official survey, and also names as granted a certain number of linear feet of a lode in that location, the latter figure has a mysterious effect upon the former. I shall endeavor to show in the present paper that this notion is without foundation; and I hope in a future paper to analyze its origin, and the nature of the support which it has received through the vagaries of the General Land Office.

4. The Revised Statutes do not authorize the distinction here made between one lode within a claim, and all the rest. The language of Section 2322 is clear. A locator must have one lode as the basis of his location; but having made the location, he has certain rights as to *all* lodes (not all *other* lodes, as Judge Hallett seems to think, and as the form of patent issued by the General Land Office implies), including the one discovered and named. And all these rights are absolutely cut off by his end-lines.

Moreover, the law does not authorize the possession by a locator, or the grant to a patentee, of any number of linear feet of a lode. It says, a *mining-claim located* (that is, a location) may equal, but shall not exceed, 1500 feet in length, along the lode. In other words, it limits the location, which is a measured tract of land. If a location claiming 1500 feet along a certain vein be so laid that it contains less than that length of the lode, the locator must submit to the loss. The Flagstaff decision settled that point forever. And in a United States patent, the words are (I quote from a patent issued in 1884; see my paper on "The Law of the Apex," *Trans.*, xii., 423):

"The United States . . . do give and grant . . . the said mining premises hereinbefore described as Lot No. ——, ... with the exclusive right of possession and enjoyment of all the land included within the exterior lines of said survey not herein expressly excepted from these presents, *and of* 1500 *linear feet of the said —— vein, lode, ledge or deposit,* for the length hereinbefore described, throughout its entire depth, although it may enter the land adjoining, and, also, of all other veins, lodes, ledges or deposits throughout their entire depth, the tops or apexes of which lie inside . . . "

The clause I have italicized is without any meaning or force, for it proposes to give a certain number of feet of the lode "for the length hereinbefore described," *i.e.*, the length of the location. The location is the conclusive limit of the grant, and the rights of the locator under it are exactly the same on all lodes having apexes within it.

5. But this decision presents still another proposition; namely, that the withdrawal of an earlier location may leave to an adjoining

later locator some right therein which otherwise he would not have had as against the location so withdrawn. That is to say, while A's location was in force upon the territory X, certain rights of B existed dormant therein, and came to life when the rights of A ceased. But, while these rights of A existed, they were "exclusive." A having received them from the United States, they revert to the United States if A surrenders or forfeits them. Let us suppose that A, instead of formally withdrawing his claim, had failed to do the annual assessment work, and thus forfeited it. Could B then assert any previously dormant right? By no means. Section 2324 distinctly says: "The claim or mine upon which such failure occurred shall be open to re-location in the same manner as if no location of the same had ever been made"—by A or B or anybody else; in other words, the ownership reverts, absolutely, to the United States. Now, what claim has B against the United States? Suppose that before A's surrender, and while A's location was in full force, B had filed a document, saying in effect: "I claim this ground in case A should ever abandon it!" Even Judge Hallett would laugh out of court such a foundation of title. But he gravely attributes importance to a much weaker foundation, namely, B's statement under oath to the government—the only mention of the territory X ever made by B in the proceedings reviewed—"I do *not* claim this ground." Can a man's disclaimer give him something which he could not get by an outright claim?

6. The question here arises: What are the lines of B's location? It seems to have been composed of two parts, X, the ground he disclaimed, and Y, the remainder. Is there any difference in effect between the grant to him of Y simply and the grant of X + Y, "excepting and excluding X?" Why should any mining locator stake out his boundaries upon another location, and thus appear to assert a claim which he immediately withdraws? What is the legal force of such boundaries established upon the land of another? The entrance upon that land for that purpose is a trespass. Can a trespasser create by his trespass a right, or the beginning of a right, which he would not have without it? In *Murphy* vs. *Cobb* (4 Col., Supp. 72), the Court held that no mining right could be acquired by surveys or locations made by mere trespassers.

To answer the foregoing questions we must understand how this absurd practice grew up. A review of its history may lead us to recognize the sole remaining excuse for its existence.

But this inquiry must be reserved for another occasion. Its pursuit would transcend the limits of this paper.

NOTE ON THE USE OF CRUDE PETROLEUM AS FUEL FOR RAISING STEAM AT THE SOUTH CHICAGO WORKS.

BY E. C. POTTER, CHICAGO, ILL.

(New York Meeting, February, 1889.)

FUEL-OIL was first substituted for coal at these works in September, 1888. It was first applied in the converting-department to the battery of boilers, consisting of 14 tubular boilers, 16 feet in length by 5 feet in diameter. To operate these boilers with coal for 24 hours required 14 stokers, 3 ash-wheelers, 6 men unloading coal, and 2 water-tenders, making a total of 25 men. With fuel-oil the same boilers require for the same period 2 water-tenders and 4 men to attend to the burners, making a total of 6 men. This gives a saving of 19 men, which, at an average of $2 per day each, amounts to $38 per day. Comparing the consumption of oil with that of coal, we find that for the week ending January 5, 1889, with an ingot output of 6403 tons, 2731 barrels of fuel-oil were used as against a necessary consumption of coal for the same work of 848 tons, showing 3.22 barrels of fuel-oil to be equivalent to one ton of Indiana block-coal. Figuring oil at 60 cents per barrel, and coal at $2.15 per ton, we have as the cost of oil $1.93, as against $2.15 for its equivalent in coal.

In December, 1888, the rail-mill battery of 26 boilers, of the same dimensions as those in the converting-department, was equipped for fuel-oil. For the week ending January 5, 1889, with a rail out-put of 5208 tons, 5987 barrels of oil were consumed, as against a necessary consumption of coal, to do the same work, of 1805 tons, showing 3.31 barrels of fuel-oil to be equivalent to one ton of Indiana block-coal, which at the costs named above would show $1.98 worth of fuel-oil to be equal to $2.15 worth of coal. The labor required to operate the rail-mill boilers for 24 hours, with coal as fuel, is: 26 stokers, 4 water-tenders, 6 ash-wheelers, 12 men unloading coal; total, 48 men. With oil as fuel, the same work requires 4 water-tenders and 6 men to attend to the burners; total, 10 men; a saving over coal of 38 men in 24 hours, which, at an average price of $2 per day, would be $76 per day.

In the brief experience we have had with oil as fuel, we find that the efficiency of the boilers is somewhat increased and the repairs are

materially lessened. The perfect cleanliness of the fuel and the ease and simplicity of supply and regulation, together with the steadiness and uniformity of steam-supply, make it, for our business at least, a most desirable substitute for coal. We have made no evaporation tests of any kind; the only test of its efficiency being the fact of its keeping our works running to their fullest capacity. This we find it will do; and this cannot be said to be true of coal, with our present boiler-capacity.

COAL VERSUS OIL IN THE PUDDLING-FURNACE AND IN RAISING STEAM.

BY G. H. BILLINGS, BOSTON, MASS.

(New York Meeting, February, 1889.)

THE following data were collected some years since in the course of a series of experiments with oil as a substitute for coal in puddling, the earlier form of the Archer apparatus being employed.

While I am not at liberty to give the name of the works at which the tests were made, I can personally warrant the accuracy of the record, and feel justified, therefore, in making this contribution to a discussion in which all trustworthy figures are likely to be helpful.

Oil in Puddling.

Starting with everything cold, we used 3518 pounds of oil to get the furnace in condition to charge a heat. After charging, we ran until we took out 13,340 pounds of puddled-blooms, using 8437 pounds, or 1333 gallons of oil. At $4\frac{7}{8}$ cents per gallon, the cost for oil per gross ton of blooms was \$10.91; with coal at \$4.87½, and a consumption of 1.5 tons per ton of blooms (the usual quantity is ton for ton), the cost for coal is \$7.31.

The quantity of oil consumed in heating the furnace to the proper temperature and to make the bottom, is not included in the oil consumption stated. The quantity used was 3518 pounds of oil. Usually, the furnace need only be lighted, but in this case it had been repaired, and it was necessary to make a new bottom. This required the quantity of oil necessary to make three heats of blooms, or 2109 pounds. Deducting this from the 3518 pounds leaves 1409 pounds of oil as the quantity consumed every week in lighting up.

In this manner, 31 cents per ton is added to the cost of puddling with oil, making the total $11.22.

Running for one week of six days, starting with everything cold, 6602 gallons of oil were used for the production of 65,595 pounds of blooms. The cost of extra labor during the week was $24, so that, at 4⅛ cents per gallon of oil, the fuel cost was $11.81 per gross ton. Coal at $4.87½, assuming a consumption of 1.5 tons per ton of blooms would cost $7.31, so that it was cheaper by $4.50 per ton of blooms. Besides, the waste heat of a puddling furnace using coal as fuel is utilized partially to generate steam. It may be estimated as equal in value to one-half the quantity generated by direct firing, or 4200 pounds of coal per turn.

Oil for Generating Steam.

On the first day, starting with everything cold, 2910 pounds of oil was used to evaporate 15,600 pounds of water, so that the cost of oil-firing was $52.81 per twenty-four hours, running night and day.

On the second day, the oil consumption, the furnace being hot, was 4156 pounds to evaporate 27,600 pounds of water. At 4⅛ cents per gallon, the oil-consumption being 656.76 gallons, the fuel cost per day of twenty-four hours was $76.82. Assuming that under the same conditions 1 pound of coal evaporated 9 pounds of water, the cost per day of twenty-four hours to evaporate 27,600 pounds of water, coal costing $4.87½ per gross ton, would be $15.96.

OIL AS A METALLURGICAL FUEL.

BY E. C. FELTON, STEELTON, PA.

(New York Meeting, February, 1889.)

AT the Pennsylvania Steel Works, Steelton, Pa., a series of trials has been made with oil as fuel in steel-heating and open-hearth steel-furnaces with the following results:

First Trial.—Hot 14-inch ingots, six to a charge, were heated in two Siemens' heating furnaces in the blooming mill, the oil being put in first in March, 1888. The oil used had been partly refined, so that the paraffine and some of the naphtha were removed. During a run of six weeks the consumption of oil was about 6½ gallons per ton of blooms, including the oil required to keep the furnace hot over Sunday.

Second Trial.—The same partly refined oil was employed for melting in a 30-ton open-hearth furnace, the charge being cold scrap and pig, with ore. Although the gas was carried from the producer to the furnace, a distance of about 300 feet, the oil-consumption, as the average of a month's run, was 48 gallons per ton of ingots, including the oil required to keep the furnace hot over Sunday.

Third Trial.—Oil was introduced on June 11th, 1888, to heat six Siemens' heating-furnaces in the blooming mill, and has since then been constantly used. For a period of six months, including the fuel-consumption on Sundays, and including the heating of some cold ingots, the consumption of Lima oil was 6 gallons per ton of blooms. The quantity of oil required naturally varies with the product of the mill. Under the most favorable circumstances—charging hot ingots and with all the stock supplied which the furnaces can handle—4½ to 5 gallons of oil are required per ton of blooms. Cold ingots must remain in the furnace for about three hours.

Fourth Trial.—Lima oil was used for heating a 30-ton open-hearth furnace, the producer being located near it. As the average of a six weeks' run, 54 gallons of oil were required per ton of ingots, including the fuel-consumption over Sundays and for starting the furnace. The record of the first week was 46.7 gallons of Lima oil per ton of ingots. It was found that the loss is somewhat greater than with coal-gas, and that some trouble was experienced from the fact that fine particles of oxidized iron clogged the checkers.

Fifth Trial.—Work in a 5-ton open-hearth furnace, in use since December, developed a fuel consumption ranging between 50 and 55 gallons of oil per ton of ingots.

Sixth Trial.—Oil was applied to the raising of steam under two 100 horse-power return-flue tubular boilers, the temperature of the feed-water being about 160° Fahrenheit. The results showed an average evaporation of about 12 pounds of water per pound of oil, the best twelve hours' work being 16 pounds of water evaporated per pound of oil. At the relative prices of oil and pea-coal, the former is not as economical under boilers as the latter.

In all of the trials the oil was vaporized in the Archer producer, an apparatus for mixing oil and superheated steam, and heating the mixture to a high temperature. From 0.5 to 0.75 pound of pea-coal is used, per gallon of oil, in the producer itself.

INDEX.

[NOTE.—In this Index, authors' names are printed in small capitals, and the titles of papers in italics. Casual reference, giving little or no information about the subject, are indicated by inclosing the page numbers in brackets. The distinction between such references and those not so marked, is, however, not always sharply defined.]

Aaron, C. H.: On loss of gold in roasting with salt, 9, 10, 14, 44; On loss of gold by volatilization, 36, 37, 44.

Abandoned mines, cultivation of mushrooms in, 248.

Actinolite in Ontario, Canada [294].

Adirondack (Lake Champlain) iron-region, N. Y. [745], 746; Iron-ores of, 721, 722, 725, 727.

Agriculture benefited by use of phosphate-slag, and by use of Spanish pyrites for making sulphuric acid [84].

Aiken, Arthur: Method of assaying and purifying gold, 7, 8 [31]; Process for refining alloys of gold, 30, 31.

Air-film attached to moving bodies [653].

Alabama coal-fields, 206; Area, 207; Available tonnage, 208; Product in 1887, 206, 207, 209, 210.

Alabama coal-mine, Warrior, Jefferson County, Ala. [214].

Alabama coke, 136, 142, 151; Sustaining power of, 147.

Alabama coke (Pratt) compared with Connellsville, 142.

Alabama-Connellsville Coal and Coke Company [223 *et seq.*].

Alabama iron-ore product in 1888, 723, 725.

Alabama iron-ores, use of large furnaces with, 135.

Alabama, manufacture and consumption of phosphoric acid fertilizer in [85].

Alabama red fossil-ores [150]; Easily reduced [149].

Alabama tar-springs [358].

Albite, the dominant feldspar in the Black Hills, Dakota [591, 595].

Aldrich, Shelby County, Ala., coal-mines at [210], 221.

Aldrich, T. H., remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 145.

Alice blast-furnace, Birmingham, Alabama [152].

Alkaline carbonates: Reaction with metallic iron, 545; With silicon, 542 *et seq.*

Alleghany Mountain in Greenbrier County, West Virginia, 118.

Allen, J. A., analysis of Muirkirk iron, 469, 470.

Aluminum in iron, method of determining, 474.

Aluminum, influence on cast-iron [473].

Aluminum-bronze stamp-mill screens, 521.

Amalgamation: Affected by the presence of manganese oxides, 776; Aided by the presence of iron, 777; In the Black Hills, 501 *et seq.*

Amalgamation-works (see Chlorination-works, Concentration-works, Gold-mills, Lixiviation-works, Reduction-works, Smelting-works and Stamp-mills).
Amendments to the Rules: Proposed, xxvii; Adopted, xl; xli.
American Institute of Mining Engineers and its mission, 485.
Ammonia in sewage, 344, 345.
Amphibole-schist in Black Hills, Dakota [498, 574].
Anaconda copper-mine, Butte, Montana [528]; Consumption of timber in, 268.
Analyses (see also Assays).
Analyses of: Asphalt: *Trinidad:* Crude, 363; Refined, 364.
Blast-furnace waste gases, 58, 59, 60, 78.
Blooms: *Pennsylvania:* Allegheny County; Pittsburgh, Carbon Iron Company, 679.
Boiler-scale, 352.
Buffalo cement, 251.
Cassiterite (see Analysis of Tin-ores).
Cast-iron (see Analysis of Pig-iron).
Cement, 251.
Cinder (see Analysis of Slag).
Coal (anthracite): *Pennsylvania:* Sullivan County; Bernice, 610, 615.
Coal (bituminous): *Alabama:* Cahaba coalfield, 63; Bibb County; Blocton, 217; Jefferson County; Coalburg, 211; Warrior, 213, 214; Woodward, 213; Walker County; Corona, 218, 219; Patton Junction, 220; *Tennessee:* 47.
Coal (graphitic): *West Virginia:* Greenbrier County, 121.
Coke: *Alabama:* Birmingham district; Bessemer, 224; Blue Creek, 154; Cahaba, 154; Coalburg, 154; Newcastle, 154; Parksville, 225; Pratt, 138, 154; Warrior, Watts Coal and Iron Company, 154, 213.
Coke-ash: *Alabama:* Birmingham district; Pratt, 138.
Columbite: *Dakota:* Black Hills, 593, 634.
Dolomite: *Alabama:* 63.
Ferro-silicon: American, Pencoat, 255; Foreign, Govan, 255.
Flue-dust: *Virginia:* Low Moor, 130.
Gas: Waste gas from blast-furnace, 58, 59, 60, 78; Water-gas, 300, 301.
Geyser-waters: *Wyoming:* Yellowstone Park; Lower Geyser Basin, Fountain Geyser, 554; Norris Geyser Basin, Fearless Geyser, 554; Upper Geyser Basin, Bee-hive Geyser, 554.
Gilsonite: *Utah:* Uintah County, 114.
Gold-ore: *Mexico:* Vera Cruz, Las Minas, 10.
Hot waters (see Analysis of Geyser-waters).
Iron (see Analysis of Blooms, Muck-bar, and Pig-iron).
Iron-ores: *Alabama:* Birmingham district (hard ore), 153; (soft ore), 153; Lakeview-Avondale (red ore), 153; Eastern Alabama (red hematite), 153; Fort Payne (hard red ore), 63; Red Mountain (hard ore and soft ore), 137; *Maryland:* "Baltimore ores" (carbonates), 464, 465, 471; Beltsville, Geo. Yokel (carbonates and hematites), 465; Branchville, Burgess, 466; G. L. Skaggs (carbonates, hematites, and limonites), 465; D. Collins's bank, 466; J. Collins's bank, 466; Contee's, J. O'Brien (carbonates and hematite), 465; Diven, 466; Jefferson, Tyson, 466; Millbrook (white ore and brown ore), 466; Montieth, 466; Walker's bank (brown ore), 466; *New Jersey:* Platt mine, 740; Morris County; Beach Glen (magnetic), 739; Mt. Hope, 740; *New York:* Hudson River carbonates: Raw, 281; Roasted, 276, 472; Clinton County; Plattsburg, 739; Essex County; Port Henry, New Bed (magnetic), 721, 739, 740,

743; Old Bed (magnetic), 739, 740, 744; Putnam County; Croton Magnetic, 737, 738; *Pennsylvania:* Lehigh Mountain, 740; Berks County; Jones, 744; *West Virginia:* Greenbrier County; Glenmore (red oxide), 120.
Limestone: *Alabama:* Jefferson County, 153; Gate City, 137; *Arizona:* Cochise County, Tombstone, 774.
Manganese-ore (argentiferous): *Arizona:* Cochise County, Tombstone, 769.
Muck-bar: *Pennsylvania:* Allegheny County; Pittsburgh, Carbon Iron Company, 679.
Nickel-ore: *Kansas:* Logan County; Russell Springs, 637.
Oyster-shells, 467, 471.
Phosphate-slag, 86 *et seq.*
Pig-iron: *Alabama:* North Birmingham (mottled and white), 64; *Maryland:* Baltimore, Stickney, 472; Prince George's County; Muirkirk, 469, 470; *New York:* Dutchess County; Amenia, Wassaic, 473; *Ohio:* Globe brand, 255, Wellston brand, 255; *Pennsylvania:* Montgomery County; Pottstown, Warwick, 127; Locality not given, 700.
Portland cement, 251.
Rails (see Analysis of Steel Rails).
Salt: Rock Salt: UNITED STATES: *Louisiana:* Iberia Parish; Petite Anse Island, Avery, 110. OTHER COUNTRIES: *Algeria:* Jeb el-Melah, 110; Ouled Kebbah, 110; *Austria:* Hall, 110; Hallstadt, 110; Wieliczka, 110; *Bavaria:* Berchtesgaden, 110; *German Lorraine:* Vic, 110; *Ireland:* Carrick-fergus, 110; *Prussia:* Stassfurt, 110; *Santo Domingo:* 110. From Wells and Lakes: UNITED STATES: *Michigan:* Saginaw, 110; *New York:* Onondaga, 110; *Ohio:* Hocking Valley, Pomeroy, 110; *Pennsylvania:* Pittsburgh, 110; *West Virginia:* Kanawha, 110. OTHER COUNTRIES: *Canada:* Ontario, Goderich, 110; *England:* Cheshire, Droitwich, 110; *German Lorraine:* Dieuze, 110.
Silver-ore: *Arizona:* Cochise County, Tombstone, 769.
Slag (see also Analysis of Phosphate-slag): *Maryland:* Prince George's County, Muirkirk, 470.
Steel: *Alabama:* North Birmingham, Henderson steel, 64.
Steel rails, 234, 237.
Tantalite: *Dakota:* Black Hills, Etta mine, 593.
Tin-ores: UNITED STATES: *California:* 595; *Dakota:* Black Hills; First Find mine (cassiterite), 595; Nigger Hill (stream-tin), 596; Occidental mine (cassiterite), 595; Southern Hills (stream-tin), 596; Tin Mountain mine (cassiterite), 595. OTHER COUNTRIES: *Bohemia:* Schlackenwald, 595; Zinnwald, 595; *Bolivia:* Tipuani, 595; *England:* Cornwall, 595, 596; *Ireland:* Wicklow, 595; *Mexico:* Xeres, 595; *Sweden:* Finbo, 595.
Uintahite: *Utah:* 114.
Waste gases from blast-furnace, 58, 59, 60, 78.
Water, unfit for use in steam-boilers, 353.
Water-gas, 300, 301.
Analysis of water for sanitary purposes, 341.
Anemometer, 66.
Animikie formation in Ontario, Canada [295, 296, 300].
Animikite in Ontario, Canada [294].
Anthracite and Coke, Separate and Mixed, in the Warwick Blast-Furnace (COOK) [xxvi], 124.
Anthracite at Bernice, Pa., overlying a bed of bituminous coal, 614, 616.
Anthracite coal-basin, Bernice, Sullivan County, Pa., 606.

Anthracite in northwestern Colorado [377].
Anthracite of Pottsville basin, Pennsylvania, thickness of, 208.
Anthracite-mines, amount of timber used in, 265.
Antiquity of man on North American continent [335].
Antwerp iron-ores, Jefferson County, New York [747].
Apatite: In the Black Hills, Dakota [592]; In Ontario, Canada [299].
Apatite, Canadian, phosphoric acid in, 87.
Apex tunnel, Aspen, Colorado [171, 175].
Appalachian coal-field, [206].
Apparatus for experiments in reduction of iron-ore, 286.
Apparatus for rapid reduction of ferric solutions in volumetric analysis, 413.
Archæan age of Black Hills ore-deposits, 571, 573, 587, 589.
Archæan formation in the Black Hills, 570, 571, 583.
Archæan granite in the Sawatch Range and at Aspen, Colorado, 161, 163, 180.
Archer gas-producer [810].
Argentiferous manganese-ores of Tombstone, Arizona, 767.
Argentite: In Ontario, Canada [294, 296]; At Rosario mine, Honduras [442].
Argillites in Black Hills, Dakota [498, 574].
Arizona: Argentiferous manganese-ores of Tombstone district, 767.
Arizona: Water-storage, 476.
Arizona: Yavapai County; Copper-deposits of Copper Basin, 479.
Arnold Hill iron-mine, Clinton County, N. Y. [747].
Arsenic determination, note on, 77.
Arsenic in Ontario, Canada [294, 298].
Arsenopyrite: In Black Hills, Dakota [498, 579, 593]; In Ontario, Canada [294].
Asbestos: In Black Hills, Dakota [498]; In Ontario, Canada [294].
Ascension theory of formation of ore-deposits [448].
Ash in Chattanooga coke, 142.
ASHBURNER, CHARLES ALBERT: *The Development and Statistics of the Alabama Coal-Fields for* 1887 [xxii], 206; *The Geology of Buffalo as Related to Natural-Gas Explorations Along the Niagara River* [xxvi], 398; On the geology of Bernice coal-basin, [607]; On petroleum and natural gas in New York state, 398, 402, 404.
Ashcroft, Colorado [159].
Ashland iron-mine, Gogebic Range, Lake Superior [719].
Aspen dolomite, Colorado: Cambrian, 168; Silurian, 168.
Aspen, Colorado: Early history, 158; Electricity applied in mining work [558], 563; Ore-deposits, discovery of, 158; Porphyry, 168; The ore an altered limestone, 204; Theory of formation of the ore, 204 *et seq.*
Aspen-Mammoth silver-mine, Aspen, Col. [170].
Aspen Mining and Smelting Company [176, 186, 189].
Aspen Mountain, Colorado: Geology and mines of, 156; General geology, 161; Geological section, 167, 168; Topographical map, 172.
Aspen silver-mine, Aspen, Col. [161, 170], 173, *et seq.*
Asphalt and Its Uses (GREENE) [xxv], 355.
Asphalt: Analysis of, 363, 364; Used in ancient times [357]; From Cuba, 362; from Utah, [115].
Asphalt Lake, Trinidad, 362, 363.
Asphalt-mastic, 361.
Asphalt pavements, 357, 366.
Asphaltene [362].

Assays: Copper, 409; Electrolytic assay of copper, 406; Gold-ores, 6, 12, 13, 14, 15, 16; Roasted gold-ores, 6, 12; Tin-ore containing columbite, 634.
Associates; Change of status to members: At Birmingham, xxi; At Buffalo, xxix; At New York, xli; Elected: At Birmingham, xxi; At Buffalo, xxviii; At New York, xl.
Atlantasaurus shales in northwestern Colorado [376].
Atlantic copper-mill, Lake Superior, Mich., cost of dressing at, 676, 677.
Atlantic copper-mine, Lake Superior, cost of mining at, 578.
Atlas works, Pittsburgh, Pa. [461, 470].
Atmospheric air, weight of cubic foot of [100].
Auriferous ores (see Gold ores).
Auriferous sulphides, chlorination of low grades of, 313.
Auriferous tin-veins in the Black Hills [589].
Aurora iron-mine, Gogebic Range, Lake Superior [719].
Austin, W. L., on silver-milling in Arizona [767].
Automatic Dumping-Cradles for Mine-Cars (MUNROE) [xlii], 564.
"Automatic" ore-feeder, Tullock [512].
Autunite in the Black Hills, Dakota [592].
Available heat of combustion of blast-furnace gases, 79.
Avery salt-mine, Petite Anse Island, Iberia Parish, Louisiana, 107 *et seq.*
Azurite: At Copper Basin, Arizona [479]; At Rosario mine, Honduras [442].

Bacillus tuberculosis in water, 346.
Baker, D., analysis of argentiferous manganese-ore, 769.
Ball-stamp in use in the Black Hills [528].
"Baltimore" iron-ores, Maryland, 463, 464.
Barber Asphalt Paving Company's works, visit to [xxix].
Barber's asphalt, or Grahamite [374].
Barium oxide in porphyry at Aspen, Colorado, 204.
Barnum iron-mine, Marquette County, Mich. [718].
Barrel-chlorination of North Carolina gold-ores, 316 *et seq.*
Baryta: In the Black Hills, Dakota [593]; In Ontario, Canada [294]; As gangue in silver-mines of Aspen, Colorado [159].
Basalt: In northwestern Colorado [377]; In Honduras, Central America [434].
Basic process [60].
Basic-slag crystals [89].
Battery (see Milling and Stamp-mills).
Bayles, J. C., estimate of fuel-consumption of Southern furnaces [140].
Beach Glen iron-mine, Morris County, N. J. [739].
Bee-hive geyser, Yellowstone Park, 451, 547, 548, 549, 554, 555.
Beekman iron-mine, Dutchess County, N. Y. [748].
Beryl in the Black Hills, Dakota [593].
Bell, Sir I. Lowthian: On blast-furnace process, [283]; On the disadvantages of soft coke in blast-furnaces [147].
Berchtesgaden salt-mine, Berchtesgaden, Bavaria [110].
Bernice anthracite coal-basin, Sullivan County, Pa., 606.
Bernice anthracite coal: Its properties and the method of mining it, 612; Analysis of, 610, 615.
Bernice breaker, peculiar crushing apparatus, 613.
Bertenshaw and McFarland bumping-tables [541].
Bessemer Coal & Coke Company, Bessemer, Ala., 225.

Bessemer iron-mine, Gogebic Range, Lake Superior [719].
Bessemer steel rails, 226 *et seq.*; Specifications for, 238.
Bestwood Colliery, Nottingham, England [430, 432].
Bibb County, Ala.: Coal-mines and seams [209], 210, 215 *et seq.*; Coal product in 1887, 207, 210.
Big Find silver-mine, Cochise County, Arizona [775].
BILLINGS, G. H., *Coal vs. Oil in the Puddling-Furnace and in Raising Steam* [xxxii], 808.
Biographical Notice of Byron W. Cheever (PETTEE) [xix].
Biographical Notice of Erich C. Schaufuss (BOWDEN) [xxxi], 419.
BIRKINBINE, JOHN: *Prominent Sources of Iron-Ore Supply* [xliii], 715; On roasting-kilns for carbonate ores [275].
BIRKINBINE, JOHN, and EDISON, THOMAS A., *The Concentration of Iron-Ore* [xxxi], 728.
Birmingham, Ala. (see also North Birmingham): Improved blast-furnace practice at, 151; Iron-ores and fuels of, 151; Meeting: May, 1888; Proceedings, xix.
Birmingham (Alabama) pig-iron, grading of, 94.
Bitite, 364; Used as an insulator [563].
Bitume factice [373].
Bitumen: Classified, 358; Defined, 357.
Bituminous cement [356].
Bituminous coal-measures in Pennsylvania, thickness of, 208.
Bituminous coal-mines, consumption of timber in, 265.
Bituminous limestones, 360, 361, 362.
Bituminous sandstones, 360.
Black Creek coal-seam, New Castle, Jefferson County, Ala. [153, 215].
Black Diamond Coal Company, Alabama [219].
Black Hills, Dakota: Geological structure, 570; Gold-ores and gold-milling, 498; Gold yield, 498, 538; Gold-deposits of Archæan age, 571, 573, 587; Homestake ore-deposit compared with the deposit at Rammelsberg, 575; Ore-deposits, 570; Ore-beds, their mode of formation, 574; Ores, and their metallurgical treatment, 588; Tin-ores, 595 *et seq.*; Yield of tin, 596.
Black Hills stamp-mills, cost of labor in, 532.
Black-tin, Analyses, 596.
Blake crusher [509, 606]; In use at Bonne Terre, Missouri [666].
Blake, Theo. A., on fine crushing of iron-ore [733].
BLAKE, WILLIAM P.; *Copper-Deposits of Copper Basin, Arizona, and their Origin* [xxxii], 479; *Note upon Some Results of the Storage of Water in Arizona* [xliii], 476; Discussion of paper by Carpenter and Headden, on the *Influence of Columbite on the Tin-Assay*, 785; On the argentiferous manganese-ores, and geology of Tombstone district, Arizona [767], 770; On the tin of the Black Hills, Dakota [589, 590]; On the effect of columbite on the metallurgy of tin, 633.
Blast-furnace fuel, 96.
Blast-furnace gases: Analyses of, 78, 79; Available heat, for firing boilers, 78.
Blast-furnace lines, 754.
Blast-furnace process: Bell's objection to soft coke, 147, 148; Discussion on economic limit of height and capacity, 135, 141 *et seq.*; Relative economy of large *vs.* small furnaces on Alabama material, 135 *et seq.*; Improved practice in the Birmingham, Ala., district, 151; Use of finely divided iron-ore, 731; With auxiliary combustion-chamber, 97.
Blast-furnaces (see also Iron-Works); UNITED STATES: *Alabama:* Bibb County;

Woodstock, Edwards [222]; Calhoun County; Anniston [222]; Colbert County; Sheffield [222]; Etowah County; Gadsden [222]; Jefferson County; Bessemer [222]; Birmingham, 143; Birmingham district; Alice [222]; Mary Pratt [152, 153, 222]; Sloss [211, 212, 222]; Williamson [222]; Ensley, 136 *et seq.*, 141 *et seq.*, 152 [222]; Oxmoor, Eureka [222]; Woodward [141, 153, 222]; *Illinois:* Cook County; South Chicago [144]; Lake County; North Chicago [756]; Chicago, Union [756, 757]; Will County; Joliet, Ethel, 285; *Maryland:* Baltimore, Stickney, 471; Prince George's County; Muirkirk, 460, 468; *Missouri:* Crawford County; Midland [756]; *New York:* Dutchess County; Amenia, Wassaic, 469, 472; *Oneida County:* Franklin [748]; Kirkland [748]; *Ohio:* Cuyahoga County; Cleveland, Cleveland Rolling Mill Company [150]; *Pennsylvania:* Allegheny County; Bessemer, Edgar Thompson, 150; Pittsburgh, 756; Isabella [150]; Lucy [150]; Bucks County; Durham [102]; Dauphin County; Steelton, Pennsylvania Steel Company [150]; Lackawanna County; Scranton [731]; Montgomery County; Pottstown, Warwick, 124, 389; *Virginia:* Alleghany County, Longdale [124]; Low Moor [124]. OTHER COUNTRIES: *Austria:* Eisenerz, Wrbna [756]; Hieflau, Ferdinand [756]; Treibach [756, 757]; *England:* Ferry Hill [143, 149]; Middlesbrough [143, 149]; Lancashire, N. Lonsdale [756]; Newport [756].

Blast-furnaces: Effect of velocity and tension of gases in, 282; Waste gas used as fuel for steam-boilers, 50.

Blende in Ontario, Canada [294].

"Block" pavements of asphalt [371].

Blocton, Bibb County, Ala.: Coal-mines and seams [209], 210, 215, [223]; Visit to [xxiii].

Blooms, analyses of, 679.

Bloom-rolls, speed of running [426].

Blue Creek coal-mines, Alabama [224]; Visit to [xxiii].

Blue Creek coke, analysis of, 154.

Blue Monday silver-mine, Cochise County, Ariz. [775].

Blue-Lead, Black Hills, 581.

Blum on phosphate-slag [89].

Bob Ingersoll tin-deposit, Black Hills, Dakota [592].

Bob's Ridge, Greenbrier County, W. Va., 116 *et seq.*

Bog iron-ore in Ontario, Canada [294].

Boiler fuel, water-gas as, 300.

Boiler incrustations: Cause, 350, 351; Preventatives, 351, 353, 354; Analyses, 352.

Boiler-scale resolvent [354].

Boiling-point of geyser waters in Yellowstone Park, 551.

Bonnybel silver-mine, Aspen, Colo., 170 *et seq.*, 202, 205; Geological section [198].

Booth, Franklin, experiments in oxidizing-roasting of Murchie pyrite, 6.

Booth, Garrett, and Blair, analyses of Wassaic ore, 472.

Boron associated with tin in the Black Hills [593].

Borronicki on phosphate-slag [89].

Boston and Arizona Smelting and Reduction Company [771].

BOWDEN, J. H., *Biographical Notice of Erich C. Schaufuss* [xxxi], 419.

Bowie, A., on the stamp-mills of the Black Hills [498].

BOWRON, WILLIAM M., *The Cost of a Ton of Pig-Iron in the Sequachee Valley* [xix], 45.

Bowen, H. C., analyses of asphalt, 363, 364.

Boyle on volatility of gold [8].

BRAINERD, ALFRED F.: *Henderson Steel* [xix], 60; *Notes on the Iron-Ores, Fuels, and Improved Blast-Furnace Practice of the Birmingham District* [xxii], 151; Analyses of Alabama coals, 213 224, 225; Of iron-ores, limestone, and cokes, 137, 138, 153, 154; On phosphorus in Alabama pig-iron [92]; Remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 141.
Brake coal-mine, Jefferson County, Ala. [214].
Brewsters iron-mine, Putnam County, N. Y. [746].
Brick-clay in Ontario, Canada [294, 297, 298].
Brierfield, Bibb County, Ala., coal-mines at, 210, 215.
Britton, J. Blodgett: Analysis of "Baltimore" iron-ore, 464; Of oyster-shells, 467; Of pig-iron, 255.
Brokenarrow, St. Clair County, Ala., coal-mines at, 210.
Broockman on phosphate-slag [89].
Brooks Locomotive Works, Dunkirk, N. Y., visit to [xxix].
Brookside, Jefferson County, Ala., coal-mines at [210, 211].
Brown, J. J., analysis of Buffalo cement, 251.
BROWNE DAVID H.: *Distribution of Phosphorus in the Ludington Mine, Iron Mountain, Michigan: A Study in Isochemic Lines* [xlii], 616; Remarks in discussion of Mr. Woodbridge's paper on *A Rapid Method for the Determination of Phosphorus in Certain Ores*, 752.
Browning, F. D., on gold chlorination in California [504].
Bruce copper-mines, Ontario, Canada [295].
Brush Electric Company's plant at Comstock mines, Nevada, 558.
Buchanan magnetic ore-separator, 737 [742].
Buffalo, New York: Geology of, 398; Asphalt pavements, 367, 369; Meeting, October, 1888; Proceedings, xxiv.
Buffalo cement, analyses of, 251.
Buffalo Cement Company, Buffalo, N. Y., 250, 398, 401, 404, 405; Visit to works [xxix].
Buffalo gypsum and cement-rock deposits, 250; Fossiliferous, 251.
Bulkley, Fred. G., remarks in discussion of Mr. Rothwell's paper on *The Present Status of Electric Transmission of Power*, 563.
Bunker Hill silver-mine, Cochise County, Ariz. [774].
Bunsen flame, colored green by chloride of gold in chlorine gas, 20, 21.
Burchard silver-mine, Cochise County, Ariz. [775].
Burden iron-mine, Columbia County, N. Y. [748].
Butte, Montana, manganese-ores, 774.
Butters, C., on loss of gold and silver in roasting, 11.

Cabarrus County, North Carolina, Phœnix gold-mine [314, 315].
Cahaba coal-field, Bibb County, Ala., 207, 209.
Cahaba Coal Mining Company, Bibb County, Ala., 215, 216.
Cahaba coke, analyses of, 154.
Calcium chloride, effect on boiling point of water, 453, note.
Calculations of the Available Heat and the Required Dimensions of Chimneys, Combustion-Chambers and Gas-Burners, in the Use of Blast-Furnace Gases for Firing Boilers (ROBERTS) [xxi], 78.
Caledonia gold-mine, Black Hills, Dakota, 576; Stamp-mill, Terraville, 500 *et seq.*, 539.
California, bituminous sandstones, 360.
Callender Insulating Company [364].

Calorific power of Lackawanna coal [303].
Cambria iron-mine, Marquette County, Mich. [718].
Cambria Iron and Steel Works, Johnstown, Cambria County, Pa. [227].
Cambrian formation: At Aspen, Colo. [157, 163, 173, 178, 179]; In the Black Hills [571].
Campbell, J. L., analysis of coal, 211.
Campbell and Raymond hot-blast stove [463].
CANBY, R. C., *Note on Arsenic Determination* [xxi], 77.
Carbon: In cast-iron changed by silicon to graphitic condition, 253; How affected by silicon, 684, 690; Graphitic, as a softener of cast-iron, 253; As an impurity in zinc [412, 418]; In steel rails [783]; Value as fuel, 99.
Carbonate iron-ores: Near Baltimore, Maryland [463]; Roasting of, on Hudson River, 275.
Carbon Hill, Walker County, Ala., coal-mines at, 210, 220.
Carbonic acid, specific heat of [100].
Carbonic oxide, specific heat of [100]; Value as fuel [99]; Weight of cubic foot of [100].
Carboniferous formation: *Arizona:* Tombstone [768]; *Colorado:* Aspen [157, 166, 174 *et seq.*, 178 *et seq.*, 205]; In northwestern Colorado, 376 *et seq.*; Red Mountain, Aspen [156]; *Dakota:* Black Hills [571]; *West Virginia:* 118.
Carboniferous limestone, the ore-bearing rock of Aspen and Leadville, Colo., 164 *et seq.*
Carbon Iron Company, Pittsburgh, Pa., 678.
Carnegie, Andrew, address of welcome at New York [xxxi].
Carnegie, D. J., on reduction of ferric-sulphate in volumetric analysis [411].
CARPENTER, FRANKLIN R., *Ore-Deposits of the Black Hills of Dakota* [xlii], 570.
CARPENTER, FRANKLIN R. AND HEADDEN., W. P., *Note on the Influence of Columbite upon the Tin-Assay* [xlii], 633.
Carrick-fergus salt-mine, Carrick-fergus, Ireland [110].
Cassel stamp-mill, Central City, Dak. [500].
Cassiterite: Analysis of, 595; In the Black Hills, 589, 590, 634.
Cast-iron (see also Pig-iron); Analysis of, 700; Silicon in, 683; Sources of its strength, 253.
Cast-steel water-jackets, 131.
Cement: Analyses, 251; Bituminous and hydraulic [356].
Cement-gravel, auriferous, in the Black Hills, 572.
Cement-Rock and Gypsum-Deposits in Buffalo (POHLMAN) [xxvi], 250.
Cement-rock: In the Black Hills [571]; At Buffalo, fossiliferous, 251; In Ontario, Canada [294, 298, 300].
Central Forge Company's works, Whitestone, Long Island, visit to [xliv].
Centrifugal-pumps: Heald & Cisco [668].
Cerussite at Rosario mine, Honduras [442].
Chalcopyrite: In Black Hills, Dakota [498, 581, 582]; In granite, Copper Basin, Arizona [482]; In Ontario, Canada [294].
"Challenge" ore-feeder, Hendy [512].
Champion iron-mine, Marquette County, Mich., 717, 729; Hand separation of ore, 729.
Champion tin-vein, Black Hills [590].
Chandler iron-mine, Vermilion Range, Minn. [719].
Change of status of associates; At Birmingham, xxi: At Buffalo, xxix; At New York, xl.

Chanute, O., experiments on crushing-strain on rails, 230.
Chapin iron-mine, Iron Mountain, Menominee County, Mich., 560, 566, 619, 718; Compressed air-plant at, 560.
Charcoal: As a precipitant for gold, 37; Used per ton of iron at Muirkirk furnace, 468 *et seq.*
Charleston rock, phosphoric acid in, 87.
Chateaugay iron-mine, Clinton County, N. Y. [747].
Chateaugay Ore and Iron Company, 722, 730, 739, 740; Ore-concentration at Lyon Mountain, N. Y., 731 [743].
"Chats," or waste-sands, and chat-tanks, 660.
Cheap fuels in blast-furnace, 96.
Chemung formation, Greenbrier County, W. Va. [117].
Cheshire salt-wells, Cheshire, England [110].
Chester Steel Castings Company, Chester, Pa. [131].
Chilling of iron affected by silicon, 705 *et seq.*
Chimney draught, head of, 80.
Chimneys, sectional area of when used with blast-furnace gases, 78 *et seq.*
Chinaman geyser, Yellowstone Park, 451, 552.
Chloride silver-mine, Aspen, Colo. [159, 170, 173, 202, 205].
Chloride of gold [7]; Test for in stream of chlorine gas, 20, 21; Higher chlorides [37].
Chloride of silver: Volatility of [8, 9]; At Rosario mine, Honduras [442].
Chloridizing muffle roasts with Murchie pyrite, 15.
Chloridizing-roasting: Losses of gold in, 9; Losses increased by increase of salt, of temperature, and of time of roasting, 14; With addition of sulphur, 776.
Chlorination of Low-Grade Auriferous Sulphides (PHILLIPS) [xxvi], 313.
Chlorination of gold-concentrates: In the Black Hills [541]; In North Carolina [541].
Chlorination, Mears's process, 315.
Chlorination-works (see also Concentration-works, Gold-mills, Lixiviation-works, Reduction-works, Smelting-works, and Stamp-mills); *California:* Grass Valley [42]; Nevada City, Maltman, 42; Merrifield mine, 42.
Chlorine: Effect of, on metals at cherry-red heat, 34; In impure water, 344; Relation to gold, 7, 19; Relative affinity for gold and the base metals [33]; Volatility of gold in, 8, 19 *et seq.*; Effect of temperature on volatility of gold in, 21 *et seq.*
CHRISTY, S. B., *The Losses in Roasting Gold-Ores and the Volatility of Gold* [xxii], 3.
Church, Prof. John A.: Concentration at Tombstone [767]; Tellurium in ores of Tombstone [771]; On sustaining power of coke [146].
Cinder: Iron saved from Bessemer converter cinder by magnetic separator, 606.
CLAGHORN, CLARENCE R., *Notes on the Bernice Anthracite Coal-Basin, Sullivan County, Pa.* [xlii], 606.
CLAPP, G. H. AND HUNT, A. E., *The Impurities of Water* [xxvi], 338.
Clapp-Griffith process [60].
Clark, Ellis, Jr., on magnetic separator at Pribram, Bohemia [736].
Clark iron-mine, St. Lawrence County, N. Y. [747].
CLARK, J. K., remarks in discussion of Mr. Goodale's paper on *The Occurrence and Treatment of the Argentiferous Manganese-Ores of Tombstone District, Arizona*, 776.
Classen on electrolytic assay of copper, 406.

Classification of coal for tariff duty on importation into Canada, 611.
Classification of ores for jigging: Is close sizing advantageous? 637 *et seq.*
Clayton, J. E., on vein-formation [449].
Cleveland iron-mine, Marquette County, Mich., 717.
Cleveland Rolling Mill Company, Cleveland, Ohio [150].
Cleveland tin-vein, Bear Gulch, Black Hills [590].
Cliff's Shaft iron-mine, Marquette County, Mich. [718].
Clifton blast-furnace, Alabama, visit to [xxiii].
"Clinkers" in roasting Hudson River ore, 278.
Clinton County, N. Y., iron-district [745], 748.
Clinton formation: In western New York [327, 328, 331, 399 *et seq.*]; In Greenbrier County, West Virginia [117]; In Ontario, Canada [300].
Clinton fossil-ores [745], 748; Hematites in New York, 748; At Red Mountain, Ala., 152.
Clove Spring iron-mine, Dutchess County, N. Y. [748].
Coal: Analyses, 47, 63, 121, 211, 213, 214, 217 *et seq.*, 610, 615; In Alabama [91]; Coal product of Alabama for 1887, 206; Miners' wages in Alabama, 222; Washing of Alabama coal before coking, 141, 145; In Cretaceous formation of northwestern Colorado, 377; In Weber shales, Aspen, Colorado, 166; In Pocono sandstone, Pennsylvania, 208; In Georgia [91]; In Tennessee, 47 [91]; Cost of mining in the Sequachee Valley, 48; In Greenbrier County, West Virginia, 119, 121; Estimated amount of bituminous coal in Pennsylvania 208; English coal sold in ports of Gulf of Mexico, 226; Coking of Alabama coals, 136.
Coal-basins (see also Coal-beds, Coal-fields, and Coal-mines); *Pennsylvania:* Pottsville, 208; Sullivan County, Bernice, 606.
Coal-beds (see also Coal-basins, Coal-fields, and Coal-mines): *Pennsylvania:* Lykens Valley, 609; Mammoth, 208.
Coal Creek coal-mines, Parksville, Ala., 225.
Coal-crusher at the Bernice breaker, 613.
Coal-fields (see also Coal-basins, Coal-beds, and Coal mines): Appalachian, 206; *Alabama:* 206; Coosa, 207 [209]; New Castle, 209; Bibb County, Cahaba [63], 207, 209; Warrior [148], 207, 209; Warrior, available tonnage of, 207; *Colorado:* 375 *et seq.; Illinois and Ohio:* [208]; *West Virginia:* [119], 121 [208]; Available tonnage of the various coal-fields in the United States, 208.
Coal-mines: UNITED STATES: *Alabama:* 210 (table); Bibb County; Brierfield Coal and Coke Company, 210, 211, 215; Cahaba Coal Mining Company, 210, 215: Jefferson County, 214; Black Creek [153, 215]; Brake [214]; Carr & Company, 210; Coalburg [152]; Henry Ellen Coal Company, 210 [223]; Hoene Consolidated Coal Company, 210, 214; Jefferson, 214; Mabel Mining Company, 210, 213; Milner Coal and Railroad Company, 210, 215 [223]; Newcastle [153], 215; New Shaft, 214; Old Shaft, 214; Pierce Warrior Coal Company, 210, 214; Parksville, Coal Creek, 225; Pratt, 152, 209 *et seq.* [223]; Sloss Iron and Steel Company, 210, 212; Tennessee Coal, Iron, and Railroad Company, 209, 210; Warrior Coal and Coke Company, 210, 211 [223]; Watts Coal and Iron Company, 210 *et seq.;* Woodward Iron Company, 210, 211, 215; Wolf Den Hollow, 214; Shelby County; Montevallo Coal and Iron Company, 210, 221 [223, 224]; St. Clair County; Coosa Coal and Coke Company, 210; St. Clair Coal Company, 210, 211, 217; Tuscaloosa County; Insane Hospital, 221; A. Durie, 210, 221; Walker County; Black Diamond Coal Company, 210, 219; Corona Coal and Coke Company, 210, 220; Day's Gap, 219; Deer Creek,

219; E. Donaldson & Co., 210; T. H. Dunn & Co., 210; Frief and Moore, 210; Kansas City Coal and Coke Company, 210, 220; B. M. Long & Co., 210; Norwell & Co., 210; O'Brien Coal Company, 210, 218; Patton, 218; Penn Mobile Coal Company, 210; Virginia and Alabama Mining and Manufacturing Company, 210, 219; Wolf Creek Coal Company, 210, 218; *Pennsylvania:* Luzerne County; Nanticoke [419]; Sullivan County; Bernice, Heiss, Jackson, and Mylert openings, 615; *Tennessee:* Bledsoe County; Ferguson [47]; Norwood [47]; Panter [47]; Seals [47]; Webb [47]; Cumberland County; Basses Creek [47]; Brown [47]; Marion County; Battle Creek [47]; Bee Branch [47]; Distribute Branch [47]; Griffith's Creek [47]; Hammet's Cove [47]; Kinnaird's Cove (outcrop) [47]; Ralston's Cove (outcrop) [47]; Victoria [47]; Whitewell [47]: Sequatchie County, Deakin's (outcrop) [47]; Heard [47]; Elliot [47]; Rankin [47]; Stone [47]; *West Virginia:* Crescent [455]; Faulkner [455]; Winifrede Coal Company [455]; Fayette County; Powellton, Mt. Carbon [454]. OTHER COUNTRIES: *England:* North Staffordshire, Sneyd [431, 432]; Nottingham, Bestwood [430, 432]; *Germany:* Westphalia, Hanover [429].

Coalburg, Jefferson County, Ala.: Coal-mines at, [141], 210, 211 [223]; Analysis of coke, 154.

Coal-slack as blast-furnace fuel [97].

Coal-tar and asphalt used together, 368.

Coal-tipples on the Kanawha River, W. Va., 455.

Coal Valley, Walker County, Ala., coal-mines at, 210, 219.

Coal Transfer of the Mt. Carbon Company, Limited (PAGE) [xlii], 454.

Coal versus *Oil in the Puddling-Furnace and in Raising Steam* (BILLINGS) [xxxii], 808.

Coke: Analysis of, 154, 224, 225; As a blast-furnace fuel, with anthracite, Warwick furnace, 124, 126; Connellsville, compared with coke from Alabama, 142; Experiments in washing Alabama coal to improve the coke, 141, 145; Objections to the use of soft coke, 147; "Retarded coke" as a fuel, 678; Amount used to ton of pig-iron in Sequachee Valley, 48; Cost of in Sequachee Valley, 48; Made from Southern coal, 136, 142; Sustaining power of Alabama coke, 147; Sustaining power of Connellsville coke, 147; Sustaining power of West Virginia coke, 147; Used in making a ton of iron in Birmingham district, 151.

Coke: Localities: *Alabama:* Blue Creek, 152, 155; Cahaba, 155; Coalburg, 155; New Castle, 155; Pratt, 155; *Tennessee:* Sequachee Valley, 47, 48; Chattanooga, 142.

Coke-ovens, Jasper, Walker County, Ala. [226].

Coking-coal of northwestern Colorado, 377 *et seq.*

Cokes of Alabama, 136; Quality of, 142; Amount produced, 211.

Colby-Davis roasting kilns for iron-ores [721].

Colby iron-mine, Gogebic Range, Lake Superior, 718.

Cold straightening of rails, 234, 785.

Coldren, Jerome, discoverer of nickel in Kansas, 636.

Color and odor of water, terms describing, 340.

Colorado: Coal-region, 375; Anthracite [377]; Coking-coal, 377; Lignite, 377; Gold-ores of Gilpin County [541]; Geology and mines of Pitkin County, 156.

Columbia County, N. Y., iron-district [745], 748.

Columbite: Analysis of, 593, 634; In the Black Hills, Dakota, 592, 633 [786]; Its influence upon the tin-assay, 633.

Combustion-chambers, dimensions when used with blast-furnace gases, 78 *et seq.*

Combustion of fuels, 99.
Comma bacillus in water [346].
Commercial value of phosphate-slag, 88.
Commonwealth iron-mine, Florence County, Wis. [718].
Comparison of high and low furnaces in iron-smelting [149].
Composition of products of combustion of blast-furnace gases, 79.
Compressed-air plant at Iron Mountain, Michigan, 560.
COMSTOCK, THEODORE B., *Hot-Spring Formations in Red Mountain District: A Reply to the Criticisms of Mr. Emmons* [xxvii], 261.
Comstock silver-mines, Nevada, electric plant at, 558.
Concentration of Iron-Ore (BIRKINBINE AND EDISON) [xxxi], 728.
Concentration of ores (see also Jigging): Iron-ores, sizing, 733; Lead-ore at Bonne Terre, Mo., 661; Tin ores in the Black Hills, 597.
Concentration-plant at Dakota School of Mines, 597.
Concentration-works (see also Chlorination-works, Gold-mills, Lixiviation-works, Reduction-works, Smelting-works, and Stamp-mills); For iron-ores: *Michigan:* Negaunee [728]; Republic [728]; *New York:* Clinton County, 730; Putnam County, 730.
Condensation of chloride of gold: From a chlorine stream, 36 *et seq.*; By aid of pyrite, 44.
Condensing-tower for removing gold from chlorine-gas, 41.
Congress mine, Red Mountain district, Colo. [264].
Conkling magnetic ore-separator, 739.
Connamara silver-mine, Aspen, Colo. [170, 173, 176, 200 *et seq.*].
Connellsville coke: Compared with coke from Alabama, 142; Sustaining power of, 147.
Contact-deposits: At Aspen, Colo., 165, 185, 186, 205, 206; In the Black Hills, 582 *et seq.*
Contact silver-mine, Cochise County, Ariz. [775].
Continental system of jigging, 637.
COOK, EDGAR S., *Anthracite and Coke, Separate and Mixed, in the Warwick Blast-Furnace* [xxvi], 124; *Tuyere Slagging-Valve* [xxvi], 389.
COOK, ROBERT ANDERSON, *Wenström Magnetic Separator* [xlii], 599.
Coosa coal-field, Alabama, 207 [209].
Copake Iron Company, strength of iron, 463.
Copper: *Arizona:* Yavapai County, Copper Basin, 479 *et seq.*; Process for treatment suggested, 484; *Dakota:* Black Hills [571]; Native, in archæan rocks, 581; *Ontario:* 294, 295, 298; Effect of chlorine on, at cherry-red heat, 35; Electrolytic assay of copper, 406, 409; In eruptive rocks [482, note]; In North Carolina gold-ores [314]; In Triassic formation of New Jersey [483].
Copper-mines: *Michigan:* Lake Superior, Atlantic, 578; *Montana:* Deer Lodge County; Anaconda [528]; *New Mexico:* Santa Rita [483]; *Ontario:* Sudbury, 295.
Copper-Deposits of Copper Basin, Arizona, and their Origin (BLAKE) [xxxii], 479.
Copper-dressing at Lake Superior [637, 640, 670]; Cost of, 676, 677.
Copper-glance: In the Black Hills [581]; In Ontario, Canada [294].
Copper-ores as cement in fragmental rocks [479].
Cordova, Walker County, Ala., coal-mines at, 210.
Corniferous limestone: Niagara, N. Y. [325, 328, 329]; Buffalo, 250, 398 *et seq.*
Cornish-rolls at Bonne Terre, Mo. [666].

Cornwall iron-ore banks, Pennsylvania, 720; Magnetic concentration at [743]; Cost of mining at [155].
Corona, Walker County, Ala., coal-mines at, 210, 218, 220.
Corundum in the Black Hills, Dakota [593].
Cost of a Ton of Pig-Iron in the Sequachee Valley (BOWRON) [xix], 45.
Cost: Of chlorinating gold concentrates, 321; Of copper-dressing at Lake Superior, 676; Of hand-picking iron-ore at Dannemora, Sweden, 604; Of lead-dressing at St. Joseph Lead Works, Bonne Terre, Mo., 676; Of milling gold in the Black Hills, 540, 577; Of mining at the Atlantic mine, Lake Superior, 578; Of mining coal in the Sequachee Valley, Tennessee, 48; Of mining iron-ore at Cornwall Bank, Lebanon, Pa. [155]; Of natural gas compared with cost of anthracite at Buffalo, 406; Of roasting gold-concentrates in North Carolina, 318; Of water-gas compared with coal as a fuel [304].
Cotta on the Rammelsberg ore-deposit [576].
Council, report of, for the year ending February, 1889, xxxiii.
Cowboy tin-mine, Black Hills, Dakota [786].
Cowles Electric Smelting and Aluminum Company [521]; Visit to works of, at Lockport, N. Y. [xxx].
Cox, E. T., analyses of cements, 251.
Crescent coal-mines, West Virginia [455].
Cresson, C. M., analysis of Bernice anthracite coal, 610.
Cretaceous formation: At Aspen, Colo. [157]; In the Black Hills [571]; In Maryland [464]; In Honduras, C. A. [434].
Cretaceous coal-measures in Colorado, 377.
Croft iron-mine, Putnam County, N. Y. [746].
Croton magnetite iron-mine, Putnam County, N. Y. [737 746]; Ores [734, 743, 744].
Crown Point iron-mine, Essex County, N. Y. [746].
Crystals in basic converter-slag [89, 90].
Crystals of chloride of gold formed in roasting, 23.
Cuban asphalt or glance-pitch, 362.
Cultivation of Mushrooms in Abandoned Mines at Akron, N. Y. (WARREN) [xxv], 248.
Cuprite in the Black Hills [581].
Custer County, Dakota, gold-ore, 578, 579.
Cycads in Honduras, C. A., 432, 435.

Daggett, Ellsworth, on the Russell process [45].
Dakota formation in northwestern Colorado [376].
Dakota: Gold-ores of the Black Hills, 498; Ore-deposits of Black Hills, 570; Tin-mines, Cowboy [786].
Dakota School of Mines, Rapid City, concentration-plant at, 597.
Dams: *Arizona:* Yavapai County, Walnut Grove, 476.
Dannemora iron-mine, Sweden, 603.
Daubrée: On copper in eruptive-rocks [482, note]; On the relation of tin-ores to other minerals, 594.
Dawes, H. F., method of phosphorus determination, 750.
Davis-Colby roasting-kilns for iron-ores [721].
Day's Gap, Walker County, Ala., coal-mines at, 210, 219.
Deadwood gold-mine, Black Hills, Dakota, 573, 574, 575, 577; Stamp-mill, 500 *et seq.*

De Bardeleben Coal and Iron Company, Ala. [152, 223 *et seq.*].
Debray: Of volatility of gold [4]; Of gold chloride [8].
Deer Creek coal-mine, Walker County, Ala., 219.
De Gournay on Koepe system of winding [432].
DELANO, FREDERIC A.: *Rail-Sections* [xxxii], 421; Quoted in discussion on steel rails, 246.
Delano, W. H.: Remarks in the discussion of Mr. Greene's paper on *Asphalt and Its Uses*, 373; On use of asphalt in foundations, 366.
DENTON, J. E., *The Economy of Modern Air-compressors* [xliii].
DENTON, J. E. and JACOBUS, D. S., *The Efficiency of a Steam-Boiler Using the Waste-Gas of a Blast-Furnace as Fuel* [xxii], 50.
De-phosphorizing iron, Thomas process [86, 92].
Deposition of hematite, 629 *et seq.*
De Smet gold-mine, Black Hills, Dakota [573, 574, 576].
Determination of Phosphorus in Iron and Steel (SHIMER) [xxi], 100.
Determination of phosphorus in certain iron-ores, 750.
Determination of Silicon in Ferro-silicons; Its Occurrence in Aluminum as Graphitoidal Silicon; and a Study of its Reactions with Alkaline Carbonates (WILLIAMS) [xxxii], 542.
Development and Statistics of the Alabama Coal-fields for 1887 (ASHBURNER) [xxii], 206.
Devereaux, W. B., on the occurrence of gold in the Black Hills [572, 573].
Deville: On silicon in aluminum [544]; On volatility of gold [4].
Devonian formation: Buffalo, N. Y., 250, 398 *et seq.*: In Ontario, Canada [294, 299]; Greenbrier County, W. Va., 116 *et seq.*; Not seen in the Black Hills [571].
DEWEY, FRED. P.: *Note on the Nickel-Ore of Russell Springs, Logan County, Kan.* [xxvi], 636; *Pig-Iron of Unusual Strength* [xxvi], 460; On sustaining power of coke [146]; On phosphate slag [89].
Dexter silver-mill, Butte, Montana [777].
Dieuze salt-wells, Dieuze, German Lorraine [110].
Differential Regenerative Hot-Blast Stove and its Application to an Open-Hearth Blast-Furnace (WAINWRIGHT) [xxvi], 132.
Discovery: Of Aspen ore-deposits, 158; Of tin in the Black Hills, Dakota, 588.
Disease germs in water, how detected, 347.
Distribution of Phosphorus in the Ludington Mine, Iron Mountain, Michigan, A Study in Isochemic Lines (BROWNE) [xlii], 616.
Dixon claim, Aspen, Colo. [178].
Dolomite: Aspen Mountain, Colo., Lower Carboniferous, 164, 165, 178; Silurian 163, 178 *et seq.*; North Birmingham, Jefferson County, Ala., 61, 63.
Dolomitization of limestone at Aspen, Colo., 164 *et seq.*, 196.
Donaldson's ore-banks, near Muirkirk, Maryland [465].
Double chloride of gold and silver supposed to be formed in Miller process, 33.
Dressing-works (see also Lead-works): St. Joseph Lead Company, Bonne Terre, Mo., 659.
Droitwich salt-wells, Droitwich, England [110].
Drown, T. M.: Analysis of gilsonite (uintahite), 114; Analyses of Wassaic pig-iron, 473; On the action of alkaline carbonates on pig-iron [543]; On reduction of ferric-sulphate in volumetric analysis [412]: Terms for describing the color and odor of surface waters, 340; Use of calcium-chloride solution in separating coal from slate [145].

Drown and Muhlenberg on method for determination of phosphorus in iron and steel [102].
Ducktown, Tenn., copper-deposit compared with copper-deposits in the Black Hills, 581.
Dumping-cradles for mine-cars, 564.
Dump-piles of iron-mines reworked [729].
Dunn iron-mine, Menominee Range, Lake Superior [718].
Durant silver-mine and incline, Aspen, Colo. [170, 189 *et seq*].
Dutchess County, N. Y., iron-district [745], 748.

Earle, F. C., analysis of limestone, 774.
Eaton on the geology of Niagara River [398].
Economy of Modern Air-Compressors (DENTON) [xliii].
Edgar Thomson Steel Company, Bessemer, Allegheny County, Pa. [150].
Edgar Thomson blast-furnace, Pittsburgh, Pa. [756].
EDISON, THOMAS A. and BIRKINBINE, JOHN, *The Concentration of Iron-Ore* [xxxi], 728.
Edison magnetic ore-separators, 741, 743.
Effect of Velocity and Tension of Gases on the Reduction of Ores in the Blast-Furnace (ROBINSON) [xxvi], 282.
Efficiency of a Steam-Boiler Using the Waste-Gas of a Blast-Furnace as Fuel (JACOBUS and DENTON) [xxii], 50.
Election of members and associates: At Birmingham, xx; At Buffalo, xxvii; At New York, xxxix.
Election of officers in February, 1889, xxxiii.
Electric plant at Comstock silver-mines, Nevada, 558.
Electric transmission of power, 555.
Electricity: Applied to mining machinery at Aspen, Colo. [156], 563; Employed in smelting and welding, 559.
Electric currents of high voltage, 561.
Electric insulation by Ferranti's system [561].
Electric motors employed in mines, 555.
Electrolytic assay of copper, 406.
Elgin decision, its bearing on the question of end-lines in mining claims, 790, 791, 794.
Elk Mountains, Colorado [180, 181].
Elliott, Andrew, description of Niagara Falls [322].
Elliott gas-analyzing instruments [51].
Elsener on volatility of gold [4].
Embolite at the Rosario mine, Honduras [442].
Emery testing-machine, Yale and Towne Manufacturing Company, Stamford, Conn., 462.
Emma silver-mining claim, Aspen, Colo. [160, 170, 176, 188 *et seq.*, 191 *et seq.*, 194, 196].
Emmerton on method for determination of phosphorus in iron and steel [102].
Emmons, S. F.: On the geological formation of the Black Hills, Dakota [572, 573]; Geology of Mosquito Range, Leadville, Colo., 167; On geology of Leadville [164]; On ore-deposits [445, 448]; On Leadville ore-deposits [586]; On ore-deposits of Red Mountain district, Colorado, 262; Replied to by Theo. B. Comstock, 261.
End-Lines and Side-Lines in the United States Mining Law (RAYMOND) [xxxi], 787.

Engineering laboratory, need of, 385.
Engineering profession and its present needs, 380.
English versus *the Continental System of Jigging—Is Close Sizing Advantageous?* (MUNROE) [xlii], 637.
English method of jigging: Its advantages, 655, 659; Its disadvantages, 656.
Ensley blast-furnaces, Ensley, Ala., 135 *et seq.* [152]; Visit to [xxii].
Enterprise Tunnel, Aspen, Colo [176].
Equal-falling spheres, diameter of, 650.
Equalization of Load on Winding-Engines by the Employment of Spiral-Drums (ROGERS) [xxvi], 305.
Erie-clay formation [402].
Eruptive rocks containing copper [482, note].
Ethel-furnace, Joliet Steel Company, Joliet, Will County, Ill. [285].
Etta tin-mine, Black Hills, Dakota [589, 591, 592, 786].
Eureka blast-furnace, Birmingham, Ala. [152].
Eureka Company [223, 224].
Eurypterus in Buffalo cement-rock, 251.
Eustis on methods of copper analysis, 406.
EVELETH, J. K., remarks in discussion of Mr. Jones's paper on *A Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis*, 757.
Experiment devised to force mercury into contact with gold, 315; To reduce iron-ore with gases of different velocity and tension, 285; To illustrate pressure on walls of blast-furnace, 148.
Experimental determination of laws governing movement of bodies in narrow channels, 640 *et seq.*
Explosion of fire-damp at Nanticoke, Pa., 419.

FACKENTHAL, B. F., JR., *Hollow Iron Pig-Patterns* [xlii], 427.
Fall of bodies in water, laws governing the, 638.
Falling-velocity of grains in water and glycerine, 653, 657.
Father de Smet stamp-mill, Central City, Dak., 500 *et seq.*
Faulkner coal-mines, West Virginia [455].
Fault-fissure at Aspen, Colo., 205.
Fearless geyser, Yellowstone Park, 554, 555.
Feasibility of Using Cheaper Fuels in the Blast-Furnace (WAINWRIGHT) [xxii], 96.
Feather River, California, electric-plant on [558].
Feed-water in jigging, and the effect of varying its quantity, 673.
Fehling on volatilization of gold by discharge of electricity [4].
Felsite in Honduras, C. A. [434].
FELTON, E. C., *Oil as a Metallurgical Fuel* [xxxii], 809.
Fentress gold-mine, Guilford County, N. C. [314].
Ferdinand blast-furnace, Hieflau, Austria [756].
FERNOW, B. E., *The Mining Industry in Its Relation to Forestry* [xxv], 264.
Ferranti system of insulation [561].
Ferric sulphate, reduction of, in volumetric analysis, 411.
Ferro-Silicon and the Economy of Its Use (KEEP and ORTON) [xxvi], 253.
Ferro-silicons: Analyses, 255; American and foreign brands compared, 254; Its influence on the properties of iron, 689 *et seq.*; Used as a softener of pig-iron, 684.
Ferry Hill blast-furnace, England [143, 149].
Fertilizers (phosphatic), annual production of, in the United States, 84, 85.

Fire-clay in the Black Hills [571].
Fire-damp explosion at Nanticoke, Pa., 419.
Firmstone, F., remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 145.
First Find tin-vein, Black Hills, Dakota [590, 595].
Fishkill iron-mine, Dutchess County, N. Y. [748].
Flagstaff decision, its bearing on the question of end-lines in mining claims, 790, 791.
Fleischer: On average composition of German slag, 86, 92; On dephosphorizing iron by the Thomas process [86, 92].
Fleming, H. S., analysis of ferro-silicon, 255.
FLEMING, H. S., KEEP W. J., and ORTON, EDWARD, JR., *Silicon in Cast-Iron* [xlii], 683.
Florence iron-mine, Florence County, Wis. [718].
Florida, manufacture and consumption of phosphoric acid fertilizer, 85.
Flue-Dust of the Furnaces at Low Moor, Va. (MEANS) [xxvi], 129.
Flue-dust from gold-roasting, not rich, 36.
Fluorine associated with tin in the Black Hills [593].
Fluor spar in Ontario, Cana. [296].
Flux of oyster shells used at Muirkirk furnace [467].
Forest of Dean iron-mine, Putnam County, N. Y. [740, 746].
Forest-management, suggestions for, 272.
Forestry, its relation to mining industry, 264.
Forrest silver-mining claim, Aspen, Colo. [170, 175, 199].
Fossil cycads in Honduras, C. A., 432, 435.
Fossil ore in New York [745], 748.
Fossils in cement-rock at Buffalo, N. Y., 251.
Foundations of asphalt-concrete for machines, 366.
Foundry-iron softened by silicon, 254.
Fountain geyser, Yellowstone Park, 549, 554, 555.
Fourth of July silver-mine, Cochise County, Ariz. [775].
France: Bituminous limestone, 360; Sandstone, 360; Tar-springs [358].
Franklinite in New Jersey [722].
Franklin Iron Works, Oneida County, N. Y. [748].
Franklin silver-mine, Aspen, Colo. [170].
Frazer, Persifor: Classification of coals [611]; Analysis of Bernice anthracite coal, 610.
Freil and Millbrook's ore-banks near Muirkirk, Md. [465].
Freire, Dr., on fever bacilli in water [346].
Fristol and Lawver, analysis of gilsonite, 114.
Fritz three-high rolling-mill [228].
Frue vanners: In use in the Black Hills, Dakota [597]; At Rosario mine, Honduras [442]; At Phœnix gold-mine, N. C. [317, 319].
Fuchs on volatility of gold [4].
Fuel: Coal *versus* Oil, 808; Combustion of, 99; Crude Petroleum used under steam-boilers, 807, 809; Feasibility of using cheaper fuels in blast-furnace, 96; "Retarded fuel" for iron-reduction, 678; Water-gas as a steam-boiler fuel, 300.
Fuel-consumption per ton of pig-iron at Warwick furnace, 124.
Fulton, John: Objection to soft coke for blast-furnaces [147]; On sustaining power of various cokes, 147.
Galena silver-mine, Aspen, Colo. [178].
Galenite: In the Black Hills, Dakota, [593]; In Ontario, Canada [294, 296].

Gangue of North Carolina gold-ores [314].
Garnet in the Black Hills, Dakota [498, 593].
Gas: Analysis of, 301; Natural gas for blast-furnaces [97]; Water-gas as a steam-boiler fuel, 300; Blast-furnace gases, analyses of, 58, 59, 60, 78; Blast-furnace gases, available heat of, for firing boilers, 78; Waste gases from blast-furnaces used as fuel for steam-boilers, 50; Effect of velocity and tension of gases on the reduction of ores in the blast-furnace, 282.
Gas-analyzing instruments, Elliott [51].
Gas-burners, required dimensions for blast-furnace gases, 78 *et seq.*
Gas-tar [358, 359].
Gate City rolling-mill, Alabama, visit to [xxiii].
Gates crusher [510].
Genth, F. A., analysis of Bernice anthracite coal, 610.
Geological formations of Ontario, Canada, 294.
Geological map of the United States, by C. H. Hitchcock [xii].
Geological section: South Park to Elk Mountain, Colorado, 162; Of well drilled at Port Colborne, Ontario, 401; At Buffalo, N. Y., 252.
Geological structure: Of the Black Hills, 570; Of Alabama coal-beds, 208; Of the Bernice anthracite coal-basin, 607, *et seq.*
Geology: Of northwestern Colorado coal-region, 376; Of Copper Basin, Yavapai County, Ariz, 479; Aspen Mountain, Colorado, 156, 161; Sawatch Range, Colorado, 156, 161; Petite Anse Island, Iberia Parish, Louisiana, 107 *et seq.*; Buffalo, N. Y., 252.
Geology of Buffalo as Related to Natural-Gas Explorations along the Niagara River (ASHBURNER) [xxvi], 398.
Georgia, manufacture and consumption of phosphoric-acid fertilizer, 85.
German system of jigging, 637.
Germania iron-mine, Gogebic Range, Lake Superior [719].
Germany, bituminous limestone, 362.
Geyserite in Yellowstone Park [548].
Geysers: Yellowstone Park, 546; Theory of action, 452.
Geyser-action accelerated by soap, 449 *et seq.*, 546 *et seq.*
Geyser-waters in Yellowstone Park; Boiling point of, 551; Temperature of [554].
Giant geyser, Yellowstone Park, 451, 549.
Giantess geyser, Yellowstone Park, 451, 452, 547, 548, 549.
Gilpin County, Colo., ores of [541].
Gilsonite from Uintah County, Utah, 113 [359].
Gjers and Grittinger roasting-kilns for iron-ores [721].
Glance-pitch or hard asphalt, 358, 359.
Glenmore Iron Estate, Greenbrier County, West Virginia (PAGE) [xxv], 115.
GLENN, WILLIAM, *Notes on the Electrolytic Assay of Copper* [xlii], 406.
"Globe" pig-iron, analysis of, 255.
Glucinum associated with tin in the Black Hills [593].
Glycerine, falling-velocity of grains in, 653.
Gmelin and Kraut on volatility of gold [3].
Goderich salt-wells, Goderich, Ontario, Canada [110].
Goesmann: On geology of Petite Anse Island [108]; On phosphate-slag [89].
Gogebic iron-range, Michigan, 716 *et seq.*, 727; Product of, 725.
Golconda silver-mine, Aspen, Colo. [170, 176, 189, 190, 196].
Gold: *Dakota:* Black Hills, 498, 571; Custer County, 578, 579; Pennington County, 578, 579, 580; At Rosario mine, Honduras [442]; In Ontario, Canada [294, 295, 296], 298.

Gold: Crystals formed in roasting, 24; Effect of heat on, 3; Relation to chlorine, 7, 19 *et seq.*; Volatility of, 3 *et seq.*; Volatility of, in atmosphere of chlorine, 8, 19; Volatilization by electric discharge [4]; Welding of gold buttons at low temperature [30, note]; In Black Hills, Archæan rock, 573; Gold product of the Black Hills in 1887, 498; Fineness of the Black Hills gold, 573; In metallic sulphides, losses in oxidizing-roasting of, 5; Gold-yield of North Carolina, 314; North Carolina auriferous sulphides, 313 *et seq.*

Gold Hill gold-mine, Rowan County, N. C. [314].

Gold Milling in the Black Hills (HOFMAN) [xlii], 498.

Gold-mills (see also Chlorination-works, Concentration-works, Lixiviation-works, Reduction-works, Smelting-works and Stamp-mills): *Dakota:* Central City, Father de Smet, 498, 500 *et seq*; *Montana:* Butte, Lexington [11].

Gold-mines: *California:* Nevada County; Murchie [3, 6, 14]; *Dakota:* Lawrence County; Golden Star [575]; Central City, De Smet [573, 574, 576]; Deadwood, Caledonia, 576; Lead City, Deadwood-Terra, 573 *et seq*; Highland [573, 577]; Homestake, 572 *et seq.*; *North Carolina:* Cabarrus County; Phœnix [314 *et seq.*, 319 *et seq.*, 541]; Guilford County; Fentress [314]; Rowan County; Gold Hill [314]; *South Carolina:* Lancaster County; Haile [314 *et seq.*]; *Honduras:* San Juancito, Rosario, 432; Trinufo [447]; Ocotillo [447]; *Mexico:* Vera Cruz, Las Minas [10].

Gold-ores: Assays of roasted ores, 6, 12, 13, 15, 16; Losses in roasting, 3; Losses in chloridizing-roasting, 9, 14, 17; Losses in roasting, effect of increased temperature on, 19; Effect of oxidizing-roasting on, 4.

Gold-placers of Black Hills, Dakota, 571.

Gold roasting-furnace at Phœnix mine, North Carolina, 317.

Golden Star gold-mine, Black Hills, Dakota, 575; Stamp-mill, 500 *et seq*, 538.

Golden Terra gold mine, Dakota, 576; Stamp-mill, 500 *et seq.*

GOODALE, CHARLES W., *The Occurrence and Treatment of the Argentiferous Manganese Ores of Tombstone District, Arizona* [xliii], 767.

GORDON, FRED. W., *Large Furnaces on Alabama Material* [xix, xxvii], 135.

Gordon-Whitwell-Cowper stoves at Ensley, Ala. [136, 137].

Goudie on mining in soft-ore bodies [104].

"Govan" ferro-silicon, analysis of, 255.

Grading of Birmingham Pig-Iron (ROBERTSON) [xxii], 94.

Graff, Matthew, inventor of "retarded fuel," 678.

Graff, William, analysis of Muirkirk pig-iron, 470.

Grahamite, or Barber's asphalt [374].

Granger water-gas [301].

Granite: Of West Aspen Mountain, 177; In Ontario, Canada, 294, 295, 298.

Granström, experiments in using fine ore in blast-furnaces, 605.

Graphite: In the Black Hills, Dakota [574, 582, 593]; In Huronian strata, Menominee County, Mich. [629]; From Cranston, R. I. [678]; In Ontario, Canada [294].

Graphite in cast-iron, and its influence on the properties of the metal, 690 *et seq.*

Graphitic carbon in cast-iron increased in amount through agency of silicon, 253.

Graphitoidal silicon, 542.

Great Britain, product of iron-ore, 715.

Green River coal-field, Colorado [377].

Greenbrier County, W. Va.: Coal, 119, 121; Iron, 115 *et seq.*; Limestone, 119, 120; Timber, 121; Geological and topographical map of, 117.

Greenbrier Mountain, West Virginia, 116 *et seq.*

GREENE, F. V., *Asphalt and Its Uses* [xxv], 355.
Greenwood iron-mine, Orange County, N. Y. [746].
Greigsville shaft, Livingston County, N. Y. [400].
Greisen associated with the tin of the Black Hills [591, 592].
Gridley, E., on the strength of iron made at Wassaic furnace, 472.
Grizzlies used in gold-mills of the Black Hills, 509.
Groddeck: On phosphate-slag [89]; On the Rammelsberg ore-deposit [576].
Guilford County, N. C., Fentress gold-mine [314].
Gunnison coal-field, Colorado [377, 378].
Guston mine, Red Mountain district, Colorado [264].
Gypsum: In the Black Hills [571]; In New Brunswick [297]; In Nova Scotia [297]; In Ontario [294, 297, 298, 299].
Gypsum and cement-rock deposits in Buffalo, 250.

Hagemann anemometer, 75, 76.
HAGUE, ARNOLD: *Soaping Geysers* [xliii], 546; On geyser-action, 450; On the leaching of rhyolite in Yellowstone Park, 448.
Haile gold-mine, Lancaster County, S. C. [314, 316, 317, 319, 320].
Hall, Prof. James, on the Geology of Niagara Falls [322, 333, 336, 399 *et seq*].
Hallock, Wm., on the boiling point of geyser-waters, 551.
Hall salt-mine, Hall, Tyrol [110].
Hallstadt salt-mine, Hallstadt, Austria [110].
Halymenites sandstone at base of coal measures in northwestern Colorado, 379.
Hamilton formation: Near Niagara River [400]; In Greenbrier County, W. Va., 116 *et seq*.
Hanauer lead-smelting works, Salt Lake City, Utah, cast-steel water-jackets used at, 131.
Hanover coal-mine, Westphalia, Germany [429, 431, 432].
HARDEN, JOHN H., *Note on the Koepe System of Winding from Shafts* [xlii], 429.
Hardness of iron, influence of silicon upon, 702.
Hardness of water, 354, 355.
Hard Up silver-mine, Cochise County, Ariz. [775].
Hare, A. W., on porphyry of Aspen Mountain, Colo. [168].
Harlem River, New York, High Bridge, 365.
Hartz jigs at Bonne-Terre, Mo., 662, 674.
Hayes, Charles E.: Experiments at chlorination works in California, 11; Investigations on losses of gold and silver in roasting gold-ores, 11, 12.
Hayes, S. Dana, analysis of Bernice anthracite coal, 610.
Hayes, Thomas, analysis of Bernice anthracite coal, 610.
Headden, W. P.: Analyses of cassiterite, 595; Analysis of columbite, 593.
HEADDEN, W. P, and CARPENTER, FRANKLIN B., *Note on the Influence of Columbite upon the Tin-Assay* [xlii], 633.
Heald & Cisco's centrifugal-pumps for elevating sand and water [668].
Heat available in blast furnace gases, 78 *et seq*.
Heavy-spar: In Ontario, Canada [296]; As gangue in ores of Aspen, Colo, 204.
Hedley on use of platinum ware in electrolytic assays [410].
Heiss coal-mine, Sullivan County, Pa. [615].
Helderberg formation: Near Niagara Falls [399]; In Greenbrier County, W. Va. [116, 117, 120].
Hematite (see also Iron-ores): *Michigan:* Iron Mountain, 617; *New York:* Jefferson County, 745; St. Lawrence County, 745; Of Lake Superior iron-region

[720]; In Ontario, Canada [294]; Brown: In Connecticut, Delaware, Maine, Maryland, Massachusetts, Missouri, Texas, Wisconsin [724]; In Southern States [723]; Fossil ore: In Southern States [723]; Magnetic properties of [736]; Theory of hematite-deposits, 629 *et seq.*

Henderson, James: Steel process, 60; Description of process, 64.

Henderson Steel (BRAINERD) [xix], 60.

Henderson steel, analyses of, 64.

Henderson Steel and Manufacturing Company, North Birmingham, Jefferson County, Ala., 60 *et seq.;* Excursion to works [xxii].

Hendy Challenge ore-feeder [512].

HENRICH, CARL, *Notes on the Geology and on Some of the Mines of Aspen Mountain, Pitkin County, Colo.* [xxii], 156.

Henry Ellen, Jefferson County, Ala., coal-mines at, 210 [223].

Heterosite in the Black Hills, Dakota [592].

HEWETT, G. C., *The Northwestern Colorado Coal Region* [xxvi], 375.

Hewitt, George, on the gold of Homestake vein, Black Hills, Dakota [573].

High Bridge, Harlem River, New York, arch of, 365.

Highland gold-mine, Black Hills, Lead City, Dak., 573; Stamp-mill, 500 *et seq.*

Highlands of the Hudson iron district [745], 746.

Hilgard on the geology of Petite Anse Island, La. [108].

Hilgenstock, G., on the condition of phosphoric acid in phosphate-slag, 89.

Hitchcock, Chas. H., geological map of the United States [xii].

Hoboken, N. J.: Session at Stevens Institute of Technology, xliii.

Hocking Valley salt-wells, Hocking Valley, Ohio [110].

HOFMAN, H. O.: *Gold-Milling in the Black Hills* [xlii], 498; On treatment of Black Hills tin-ores, 597.

Holley, Alexander Lyman, 227 *et seq.;* Memorial of [xi, xii].

Hollow Iron Pig-Patterns (FACKENTHAL) [xlii], 427.

Holly Manufacturing Company's Works, Lockport, N. Y., visit to [xxx].

Homberg: On loss of gold in roasting pyrites [4]; On volatilization of gold [4, 7].

Homestake gold-mine, Black Hills, Lead City, Dak., 572 *et seq.;* Stamp-mill, 500 *et seq.*, 538.

Honduras, Rosario gold-mine at San Juancito, 432.

Honorary Members, list of, ix.

Horse Creek mines, Walker County, Ala., coal-mines at, 210.

Hot-blast stoves: Raymond and Campbell [463]; Regenerative, 680; Differential regenerative, 132.

Hot-Spring Formations in Red Mountain District, Colorado: A Reply to the Criticisms of Mr. Emmons (COMSTOCK) [xxvii], 261.

Hot-Springs in San Juan district, Colorado, 262.

Hot straightening of rails, 234, 245.

Howe, H. M., on the effect of silicon upon cast-iron [685].

Howell and White furnace [771].

Hudson River carbonates, roasting of, 275.

Hudson River, New York, iron-district [745], 748.

Hudson River formation, near Niagara River [402].

Hughes, Herbert W., discussion of Mr. Harden's *Note on the Koepe System of Winding from Shafts*, 431.

Humboldt, Mich., iron-ore concentration at, 743.

Humboldt iron-mine, Marquette County, Mich. [718].

HUNGERFORD, W. S., *Mining in Soft Ore-Bodies at Low Moor* [xxii], 103.

HUNT, ALFRED E., *A Note upon a Modification of the Reducing Process Used by the Carbon Iron Company* [xliii], 678.
HUNT, A. E. and CLAPP, G. H., *The Impurities of Water* [xxvi], 338.
HUNT, ROBERT W.: *Steel Rails and Specifications for their Manufacture* [xxv], 226; *Proposed Rail-Sections* [xxxii], 778.
Hunt, T. Sterry, on granitic-veins of the Laurentian formation [594].
Huntilite in Ontario, Canada [294].
Huronian and Laurentian series of Archæan period, the ore-bearing rock of Ontario, 295.
Huronian formation: At Iron Mountain, Michigan, 617; Contains graphite, 629; In Ontario, Canada [295, 300].
Hutchinson Incline, Aspen, Colo. [194].
Hutch-work in jigging, and its enrichment, 656.
Hydraulic cement [356]; In Ontario, Canada [294, 298, 300].
Hydrocarbons (liquid), value as fuel, 99.
Hydrogen: Specific heat of [100]; Value as fuel, 99; Weight of cubic foot of [100].

Iles, M. W., analysis of silver-ore, 769.
Iletsk salt-quarry, Iletsk, Western Siberia [111].
Illinois, available tonnage of the coal-fields of, 208.
Ilmenite in the Black Hills, Dakota [582, 593, 786].
Impregnations of limestone at Aspen, Colo., 206.
Impurities of Water (HUNT and CLAPP) [xxvi], 338.
Independence silver-mine, Cochise County, Ariz. [775].
Influence of columbite upon the tin-assay, 633.
Inman, A. L., on concentration of Chateaugay iron-ores, 731.
Insulation of electric currents, Ferranti system [561].
Interstitial channels formed in jigging, their influence, 651, 672.
Isochemic lines, 616, 622; Relation to physical phenomena at the Ludington mine, 627.
Inspection of rails, 240, 244, 245.
Iron (see also Cast-iron, Pig-iron, and Steel). In the Black Hills [571]; At Muirkirk, Md., 460; Silicon-content of Ohio iron, 254; In Ontario, Canada, [294, 298, 299]; White: Analysis of, 87; Its action in assisting amalgamation, 777; Cost of making pig-iron in Greenbrier County, W. Va., 123; De-phosphorizing, by Thomas process [86, 92]; Determination of phosphorus in, 100; Effect of chlorine on at cherry-red heat, 35; Production of various countries, 715; Its several properties affected by silicon in various ways, 683 *et seq.*; Pig-iron of unusual strength, 460.
Iron Hill silver-lead mine, Black Hills, Dakota [584].
Iron King iron-mine, Gogebic Range, Lake Superior, 719.
Iron-mines: *Maryland:* Collins's Banks [466]; Diven [466]; Millbrook [466]; Montieth [466]; Walker Bank [466]; Jefferson, Tyson [466]; Prince George's County; Beltsville, Yokel's [465]; Branchville, Burgess [466]; Skaggs [465]; Contee's, O'Brien [465]; Muirkirk [465]; *Michigan:* Marquette County; Barnum [718]; Cambria [718]; Champion, 717, 729; Cleveland, 717; Cliff's Shaft [718]; Humboldt [718]; Jackson, 717; Lake Angeline, 717 [753]; Lake Superior, 717; Michigamme [718]; New York [718]; Pittsburgh and Lake Superior [718]; Republic, 717; Salisbury [718]; Winthrop [718]; *Michigan and Wisconsin:* Gogebic Range; Ashland [719]; Aurora [719]; Bes-

semer [719]; Colby, 718; Germania [719]; Iron King [719]; Ironton [719]; Kakagon [719]; Nimikon [719]; Norrie, 718; Odanah [719]; Puritan [719]; Sunday Lake [719]; Superior [719]; Menominee Range; Chapin, 560, 566, 619, 718; Commonwealth [718]; Dunn [718]; Florence [718]; Iron River 629, 718]; Ludington, 560, 616 *et seq.*, 718; Mastodon [718]; Nanaimo [629]; Norway, 718; Paint River [629]; Perkins [718]; Vulcan [718]; *Minnesota:* Vermilion Range; Chandler [719]; *Missouri:* Iron County; Pilot Knob [723]; St. Francis County; Iron Mountain [723, 730]; *New Jersey:* Platt [740]; Morris County, Beach Glen [739]; Mt. Hope [740]; *New York:* Clinton County; Arnold Hill [747]; Chateaugay Ore and Iron Company [722, 730, 739, 740, 747]; Bunker Hill [747]; Columbia County; Burden [748]; Weed [748]; Dutchess County; Beekman [748]; Clove [748]; Fishkill [748]; Essex County; Crown Point [746]; New Bed [721, 739, 740, 744]; Old Bed [739, 740, 744]; Port Henry [746, 747]; Jefferson County; Keene [747]; Shirtliff [747]; Orange County; Greenwood [746]; Putnam County, Brewster's [746]; Croft [746]; Croton Magnetite [737, 746]; Forest of Dean [746]; Mahopac [746]; McCollum [746]; Theal [746]; Tilly Foster [746], 758 *et seq.*; Todd [746]; Rockland County; Sterling [746]; St. Lawrence County, Clark [747]; Jayville [747]; Pike [747]; Polley [747]; Tate [747]; *Pennsylvania:* Berks County; Jones [744]; *Virginia:* Alleghany County; Low Moor, 103 *et seq.*; *Sweden:* Dannemora [603].

Iron-mines of the United States, consumption of timber in, 267.

Iron-mining industry of New York for past decade, 745.

Iron Mountain, Michigan, Ludington iron-mine, 616.

Iron Mountain iron-mine, Missouri [723, 730].

Iron-ores: Analyses (see Analyses of Iron-ores).

Iron-ores: In Alabama [91]; In Colorado [724]; In Georgia [91]; In Oregon [724]; In Tennessee [91]; In Washington Territory [724]; Prominent sources of supply, 715; Extent of importations from foreign countries, 1879–1888, 715, 724, 727; Cost in Sequachee Valley, Tennessee, 46; Determination of phosphorus in, 750; Phosphatic ores in the Southern States [85]; Use of finely-divided ores in the blast-furnace, 731; Concentration of iron-ores, 728; Sizing in concentration, 733; Concentration with aid of magnetic separators, 599.

Iron-ores: Localities; *Alabama:* Red Mountain (fossil), 137, 152; Chilton County (phosphoritic, metamorphic) [91, 93]; Jefferson County; Birmingham, 151; Talladega County (silurian, brown) [91, 93]; *Maryland:* "Baltimore Ores" (carbonates, altered and unaltered), 464; Muirkirk ores, 466; *Michigan:* Menominee County (blue hematite), 617, 619; *New Jersey:* 722; *New York:* Adirondack and Lake Champlain region (magnetite), 721, 728 [745], 746; Highlands of the Hudson (magnetites) [745], 746; Hudson River (carbonates) [745], 748; Clinton County (fossil ores) [745], 748; Columbia County (limonite) [745], 748; Dutchess County (limonite) [745], 748; Wassaic, 472; Essex County; Mineville (magnetite), 750; Jefferson County (red hematite) [745], 747; Putnam County (magnetite), 737; Richmond County; Staten Island (limonite) [745], 749; St. Lawrence County (red hematite) [745], 747; Wayne County (fossil ores) [745], 748; *Pennsylvania:* Cornwall ore-banks, 720; *West Virginia:* Greenbrier County; Glenmore (red fossil), 115, 119.

Iron pig-patterns, 427.

Iron-pyrites in Black Hills, Dakota [498].

Iron River iron-mine, Menominee Range, Lake Superior [629, 718].

Ironton iron-mine, Gogebic Range, Lake Superior [719].
Iron-works (see also Blast-furnaces): UNITED STATES: *Alabama:* Jefferson County; Birmingham, Pioneer Mining and Manufacturing Company [143]; Sloss Iron and Steel Company, 211, 212; North Birmingham, Sloss Iron and Steel Company, 61; Woodward [141]; *Maryland:* Baltimore, Stickney, 471; *Massachusetts:* South Boston [460]; *New York:* Oneida County; Franklin [748]; Kirkland [748]; *Pennsylvania:* Allegheny County; Pittsburgh, Carbon Iron Company, 678; Bucks County; Durham [102]; Cambria County; Johnstown, Cambria, 227; Montgomery County; Pottstown, Warwick [124]; *Virginia:* Alleghany County; Longdale [124]; Low Moor [123, 124, 129, 130]. OTHER COUNTRIES: *Germany:* Aachen, Rothe Erde [93]; Ruhrort, Rhein Steel Works [93]; *Lorraine:* Hayingen, Fois de Wendel and Company [93].
Irving, R. D., on the origin of the iron-ore beds of Menominee County, Mich., 629.
Isabella furnace, Pittsburgh, Pa. [150].

Jackson, Prof. Wendell, microscopical examination of porphyry of Aspen Mountain, Colorado, 168.
Jackson coal-mine, Sullivan County, Pa. [615].
Jackson iron-mine, Marquette County, Mich., 717.
JACOBUS, D. S., *Water-Gas as a Steam-Boiler Fuel* [xxvi.], 300.
JACOBUS, D. S., and DENTON, J. E., *The Efficiency of a Steam-Boiler Using the Waste-Gas of a Blast-Furnace as Fuel* [xxii], 50.
Jagger coal-bed, Alabama, 221.
Jamme, George, remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 141.
Jasper, its influence on the phosphorus-content of ore in Ludington mine, 625, 627.
Jayville iron-mines, St. Lawrence County, N. Y. [747].
Jebel-Melah salt-mine, Jebel-Melah, Algeria [110].
Jefferson coal-mine, Alabama [214].
Jefferson County, Ala.: Coal-mines of, 210 *et seq.*; Product of coal in 1887, 207, 210 *et seq.*
Jefferson County, N. Y., iron district [745], 747.
Jenney on structure of tin-bearing veins of the Black Hills, Dakota [590].
Jesse silver-mine, Aspen, Colo. [170].
Jigging, English system *versus* the Continental, 637.
Jigging: Theory of, 638; Effect of varying the velocity of the rising current of water, 647; Advantages and disadvantages of English method, 655, 656; In Lake Superior copper-mills, 637, 640, 670; In St. Joseph Lead Company's dressing-works, 637, 654, 661, 669.
Jigs: Hartz jig, 662, 674; Parsons jig, 661, 662, 669.
Jig-sieves, the effect of size of mesh on the jigging process, 652, 658.
JONES, CLEMENS, *A Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis* [xlii], 411; Discussion of paper, 757.
Jones iron-mine, Berks County, Pa. [744].
Jurassic formation: At Aspen, Colo. [157]; In the Black Hills [571]; In Maryland [464]; In Honduras, C. A. [438].
Jura-trias formation in northwestern Colorado [376].

Kakagon iron-mine, Gogebic Range, Lake Superior [719].
Kanawha salt-wells, Kanawha, W. Va. [110].
Kansas, nickel-ore of Russell Springs, 636.

Kansas City Coal and Coke Company, Alabama [220].
Kebler, E. A., analysis of iron, 700.
Keene iron-mine, Jefferson County, N. Y. [747].
Keep, W. J., on the influence of aluminum on cast-iron [473].
KEEP, W. J., FLEMING, H. S., and ORTON, EDWARD, JR.: *Silicon in Cast-Iron* [xlii], 683; Methods of testing the influence of silicon upon iron, 686.
KEEP, W. J., and ORTON, EDWARD, JR., *Ferro-Silicon, and the Economy of Its Use* [xxvi], 253.
"Keep's Tests" [257, 474, 686].
Kentucky: Iron from lean ores used as "softeners," 254; Tar-springs [358].
Kiddie on use of platinum ware in electrolytic assays [410].
King *et al, versus* the Amy and Silversmith Mining Company, features of case, 787 *et seq.*
Kirkland blast-furnace, Oneida County, N. Y. [748].
Knoxville silver-mine, Cochise County, Ariz., 767 *et seq.*
Koch on bacillus in water, 346.
Koepe system of winding from shafts, 429.
Kraut on volatility of gold [3, 7].
Kunhardt and Maynard on iron-ore concentration at Lyon Mountain [732, 734].
Küstel on loss of gold by volatilization, 5, 9.

Labor, cost of, in Sequachee Valley, Tenn., 49.
Labor and wages in the Black Hills, 532.
Laccolites in the Black Hills [571, 572].
Lackawanna Iron and Coal Company, Scranton, Pa. [606, 731, 732].
Lake Angeline iron-mine, Marquette County, Mich., 717 [753].
Lake Champlain iron region, N. Y. [745], 746; Mines, 721; Ores, 721, 722, 725, 727.
Lake Erie, pre-glacial valley of, 322 *et seq.*
Lake Superior Iron-Region, 716, 717, 725, 727.
Lake Superior iron-mine, Marquette County, Mich., 717.
Lakes, Prof., on the geology of Aspen mining region, 165 *et seq.*, 204.
Lancaster County, S. C., Haile gold-mine [314].
Landis, E. K., analysis of Croton iron-ore, 737, 744.
Large Furnaces on Alabama Material (GORDON) [xix, xxvii], 135.
La Salle silver-mine and claim, Aspen, Colo. [170, 173, 175].
Las Minas gold-mine, State of Vera Cruz, Mexico [10].
Last Scheme silver-mine, Aspen, Pitkin County, Colo. [167].
Late Acquisition silver-mine, Aspen, Colo. [178].
Laurentian formation in Ontario, Canada [294, 295].
Lawrence County, Dakota, stamp-mills, 498.
Lawver and Fristol, analysis of gilsonite, 114.
Leaching theory of formation of ore-deposits [445, 448].
Lead-silver ores in the Black Hills, 582.
Lead in Ontario, Canada [294].
Lead dressing-works, Missouri, 564.
Lead, effect of chlorine on, at cherry-red heat, 36.
Lead-poisoning caused by small quantities of lead in water, 346.
Leadville ore-deposits and Black Hill deposits compared, 586.
Lead-works: *Missouri:* St. François County; Bonne-Terre, St. Joseph, 637.
Le Conte, Joseph, on deposition of ore by leaching from walls [445].

LEGGETT, THOMAS H., *Notes on the Rosario Mine at San Juancito, Honduras, C. A.*, [xxvi], 432.
Legrand on boiling point of saline solutions [453, note].
Lexington mill (lixiviating), Butte, Mont. [11].
Life-History of Niagara (POHLMAN) [xxv], 322.
Lignite: In Ontario, Canada [294]; In Northwestern Colorado [377]; On Petite Anse Island [107].
Limestone: *Colorado:* Aspen Mountain, 163, 174, 178 *et seq.;* In Greenbrier County, W. Va., 118, 120; In Ontario, Canada, 294, 298; Analyses, 153, 774.
Limonite: In New York [745], 748 [749]; At Rosario mine, Honduras [442] (see also Hematite, brown).
Lindley, Douglas, experiments in oxidizing-roasting of Murchie pyrite, 6.
Lithium associated with tin in the Black Hills [593].
Lithographic-stone in Ontario, Canada [294]
Little Giant silver-mine, Aspen, Colo. [170, 171, 175, 179].
Little River iron-mine, St. Lawrence County, N. Y. [740, 747].
Lixiviation-works (see also Chlorination-works, Concentration-works, Gold-mills, Reduction-works, Smelting-works, and Stamp-mills). *Montana:* Butte, Lexington [11].
Llewellyn, N. J., Session at the Edison Laboratory, xxxi.
Locke, J. M., on gilsonite [113].
Logan County, Kansas, nickel ore, 636.
Long Island, magnetic sands [735, 737].
Longdale, Va.: Method of mining ore-body [103]; Longdale iron-works [124].
Longfellow silver-mine, Cochise County, Ariz. [775].
Lord, Prof. N. W., analysis of ferro-silicon, 255.
Loss in jigging at Bonne-Terre, Mo., and at Lake Superior, 676.
Loss of gold in chloridizing-roasting, 10, 12, 13, 43, 44; Effect of time on losses in chloridizing-roasting, 15, 16, 17; Effect of rapid evolution of chlorine, 17; Effect of temperature on, 18 *et seq.*
Loss of mercury in the Black Hills stamp-mills, 530.
Losses in Roasting Gold-Ores and the Volatility of Gold (CHRISTY) [xxii], 3.
Losses in roasting gold-ores, conditions for reducing to a minimum, 42.
Louisiana, manufacture and consumption of phosphoric acid fertilizer, 85.
Low, A. H., analysis of silver-ore, 769.
Lower Geyser Basin, Yellowstone Park, 547, 549, 554.
Low Moor, Alleghany County, Va.: Iron-mine, 103 *et seq.;* Mining in soft ore-body, 103; Furnaces, flue-dust, 129 *et seq.*
Low Moor iron-works, Low Moor, Va. [123, 124, 129, 130].
Loyalsock-Mahoopany coal-fields, Pa. [607].
Luck Now silver-mine, Cochise County, Ariz. [775].
Luck Sure silver-mine, Cochise County, Ariz. [767, 774, 775].
Lucky Cuss silver-mine, Cochise County, Ariz. [767, 774, 775].
Lucy furnace, Pittsburgh, Pa. [150].
Ludington iron-mine, Iron Mountain, Mich., 560, 616 *et seq.*, 718.
Ludington iron-ore deposit: Its physical features, 619 *et seq.;* Theory of its formation, 627.
Lumberman's Mining Company, Iron Mountain, Mich. [616].
Lupton, N. T., analyses of Alabama coals, 213.
Lyell on geology of Niagara Falls [324, 333].
Lykens Valley coal-bed, Pennsylvania, 609, 611 note.
Lyon Mountain, iron-ore concentration at [732, 743].

Mabel Mining Company, Alabama [213].
Mabery on the influence of aluminum on cast-iron, 473.
Mackintosh on use of sulphuric acid in electrolytic assay of copper [408].
Magnesian limestone (see also Dolomite): At Bonne-Terre, Mo., 661.
Magnetic ore-separators, 599, 735 *et seq.*
Magnetite: Its composition and properties, 736; In Lake Superior iron-region [720, 721]; In Eastern New York [724]; In Highlands of Hudson River, 745 *et seq.*; In Adirondack region, 745; In New Jersey [722]; In Ontario, Canada [294].
Mahlet on Koepe system of winding [432].
Mahopac iron-mine, Putnam County, N. Y. [746].
Malachite: In the Black Hills [581]; At Copper Basin, Arizona, [479 *et seq.*]; In Ontario, Canada [294]; At Rosario mine, Honduras [442].
Malo, Leon: On asphalt, 356, 358; Nomenclature of asphalt and bitumen, 373, 374.
Maltman chlorination-works, Nevada City, Cal. [42].
Maltzan on phosphate-slag [89].
Mammoth coal-bed, Pottsville, Pa., thickness of, 208.
Mammoth silver-mine, Cochise County, Ariz. [774].
Manby, C. B., analysis of Greenbrier iron-ore, 120.
Manganese: In foreign ferro-silicon [256]; In steel rails [783].
Manganese-ores: *Arizona:* Tombstone district, 767.
Manganese oxides, their influence on amalgamation, 776.
Manganite at Rosario mine, Honduras [442].
Manometer, water, construction and use of, 66 *et seq.*
Mansfield, G. W., on the electric motor in mining operations, 555, 556, 557.
Marble in Ontario, Canada [294 *et seq.*]
Marcet on phenomena of superheated waters [551].
Margaret tin-vein, Black Hills [590].
Marguerite's method for reduction of ferric sulphate in volumetric analysis, 411, 412.
Marl in Ontario, Canada [294].
Marquette iron-range, Michigan: Mines of, 717, 718; Product, 716, 717, 725, 727.
Marsh gas: Value as fuel, 99; Weight of cubic foot of [100].
Mary Murphy silver-mine, Saint Elmo, Chaffee County, Colo. [159].
Mary Pratt furnace, Birmingham, Ala. [152, 153].
Martite, feebly magnetic [736].
Mastic asphalt, 361, 366, 373.
Mastodon bones found near Petite Anse Island [107].
Mastodon iron-mine, Menominee range, Lake Superior [718].
Mathez, Auguste, remarks in discussion of Mr. Hofman's paper on *Gold-Milling in the Black Hills*, 541.
Mattes, C. C., system of measuring open-pit, 765.
Mauch Chunk formation in Bernice coal-basin, 607.
Maynard and Kunhardt on iron-ore concentration at Lyon Mountain [732, 734].
Maynard, G. W., on deposition of copper on organic remains in the Urals [483].
McCalley, Henry, estimate of area of Alabama coal-fields, 207.
McCann silver-mine, Cochise County, Ariz. [775].
McCollum iron-mine, Putnam County, N. Y. [746].
McCreath, Andrew, analysis of Bernice anthracite, 610, 615.
McDowell, F. H., *The Reopening of the Tilly Foster Iron-Mine* [xliii], 758.
McFarland & Bertenshaw bumping-tables [541].

McMillin, Emerson, analysis of water-gas, 301.
MEANS, ELLISON C., *The Flue-Dust of the Furnaces at Low Moor, Virginia* [xxvi], 129.
Mears's chlorination process, 315.
Medina formation: *New York:* Near Buffalo, 398 *et seq.;* Near Niagara [327, 328, 331, 332]; *West Virginia:* Greenbrier County [117, 118].
Meetings of the Institute from 1871 to 1889, x; At Birmingham, Ala., May, 1888, Proceedings, xix; At Buffalo, N. Y., October, 1888, Proceedings, xxiv; At New York, N. Y., February, 1889, Proceedings, xxxi.
Members and associates elected: At Birmingham, xx; At Buffalo, xxvii; At New York, xxxix.
Menaccanite, feebly magnetic [736].
Menominee County, Mich., and its iron-ores, 619 *et seq.*
Menominee iron-range, Michigan: Iron-mines, 718; Product, 716, 717, 725, 727.
Mercury forced into contact with gold [315].
Mercury-traps in use in the Black Hills stamp-mills [526, 527, 538].
Merrifield mine, Nevada City, Cal., chlorination works [42].
Merrill, F. J. H., on the rhyolite at the Rosario mine, Honduras [438].
MERRITT, WILLIAM HAMILTON, *The Minerals of Ontario and Their Development* [xxvi], 293.
Mesozoic formations in the Black Hills [572].
Metasomasis of limestone at Aspen, Colo., 204.
Metallic poisons in potable waters, 345.
Metallurgical treatment of Black Hills gold-ores, 588.
Metals in water, simple test for, 346.
Metcalf, William, on steel [237].
Methods of lead-mining at Bonne-Terre, Mo., 661.
Methods of mining in soft ore-bodies, 103.
Methods of mining in the Black Hills, 576.
Mexico: Glance-pitch, 359; Bituminous limestones, 362.
Mica in Ontario, Canada [294, 298].
Mica-schist in Black Hills, Dakota [498].
Michigamme iron-mine, Marquette County, Mich. [718].
Michigan iron-mines, 716, 717, 718, 719.
Microscopic examination of water, 347.
Middlesbrough blast-furnace and steel works, England [86, 143, 149].
Midland blast-furnace, Crawford County, Mo. [756].
Millinee silver-mine, Aspen, Colo. [170].
Milling: Gold-milling in the Black Hills, Dakota, 498.
Miller: Chlorination process [7]; Process for refining alloys of gold, 30 *et seq.*
Milner Coal, Iron, and Railroad Company, New Castle, Ala., 153.
Mineral oil [357].
Mineral pitch [357].
Mineral production of Ontario, Canada, 298.
Mineral waters, general composition of, 349.
Minerals of Ontario and Their Development (MERRITT) [xxvi], 293.
Mines, abandoned, cultivation of mushrooms in, 248.
Mines of Aspen Mountain, Colorado, 156.
Mine-timbering: Best woods for, 269; Consumption of wood in, 265 *et seq.;* Time for felling timber, 270.
Mineville iron-mines, Essex County, N. Y. [722].

Mining Industry in Its Relation to Forestry (FERNOW) [xxv], 264.
Mining in Soft Ore-Bodies at Low Moor (HUNGERFORD) [xxii], 103.
Mining at Petite Anse Island, 107 *et seq.*
Mining law, end-lines and side-lines, 787 *et seq.*
Minnesota Iron Company [719].
Minnesota: Iron-mines of Vermilion district, 719; Product, 716, 717.
Miocene formation in the Black Hills [571].
Mispickel in Ontario, Canada [294, 298].
Mississippi, manufacture and consumption of phosphoric acid fertilizer, 85.
Missouri: Iron-mines, 723; Iron-ore product, 723, 725, 727.
Missouri Boy silver-mine, Aspen, Colo. [178].
Moffat, E. S., on use of fine ore in blast-furnaces, 731, 732.
Monarch geyser, Yellowstone Park, 550.
Monarch magnetic ore-separator, 740.
Monarch silver-mine, Aspen, Colo. [178].
Mont Alto, Pa., magnetic concentration of iron-ores at [743].
Montana, manganese-ores at Butte, 774.
Moore, G. E., analysis of water-gas, 300.
Moreau, G., composition of Teplitz and Witkowitz phosphate-slag, 87, 93.
Morris iron-mines, Red Mountain, Alabama, visit to [xxii].
Morrow's mines, Jefferson County, Ala., coal-mines at, 210.
Mortars in stamp-batteries, Black Hills, 515 *et seq.*
Mosquito Range, Leadville, Colo., geology of, 167.
Moulton silver-mill, Butte, Mon. [776, 777].
Mount Massive, Sawatch Range, Colorado [161].
Mt. Carbon Coal Company's coal transfer, 454.
Mt. Hope iron-mines, Morris County, N. J. [740].
Muhlenberg and Drown on method for determination of phosphorus in iron and steel [102].
Muirkirk blast-furnace, Prince George's County, Md., 460; Record of, 468.
Munro on phosphate-slag [89].
MUNROE, H. S.: *Automatic Dumping-Cradles for Mine-Cars* [xlii], 564; *English* versus *the Continental System of Jigging—Is Close Sizing Advantageous?* [xlii], 637; *New Dressing-Works of the St. Joseph Lead Company, at Bonne-Terre, Missouri* [xxvii], 659.
Murchie gold and silver-mine, Nevada County, Cal., [3] [6] [14]; Oxidizing-roasting of pyrite from, 6.
Murchison on the deposition of copper on organic remains in the Urals [483].
Murphree's Valley, Alabama, coal and iron-fields of [225].
Mushrooms cultivated in abandoned mines, 248.
Mylert coal-mine, Sullivan County, Pa. [615].

Nails from Tin-Scrap (SMITH) [xliii], 495.
Nanaimo iron-mine, Menominee Range, Lake Superior [629].
Nanticoke, Pa., gas explosion, 419.
Natural gas; At Getzville, N. Y., 403; At Tonawanda, N. Y., 403; At Port Colborne, Ontario, 401, 402.
Natural gas: Explorations along Niagara River, 398, 402; As blast-furnace fuel [97].
Navassa rock, phosphoric acid in, 87.
Negaunee, Mich., iron-ore concentration-works at [728].

Nepigon formation in Ontario, Canada [295].
"Nessler" process for determining strength of dilute solutions of ammonia, 345.
New Bed iron-mine, Essex County, N. Y. [721, 739, 740, 743].
Newberry, Prof. J. S.: On fossil cycads in Honduras, 435; On geology of Niagara Falls [322, 325].
New Castle, Jefferson County, Ala., coal-mines at, 153, 209, 210, 215 [223].
New Castle coke, analyses of, 154.
New Dressing-Works of the St. Joseph Lead Company, at Bonne-Terre, Missouri (MUNROE) [xxvii], 659.
New Jersey: Iron-mines, 722; Product of iron-ore, 722, 725, 727.
Newport blast-furnace, Newport, England [756].
New River coals, Greenbrier County, W. Va. [119, 121], 122.
New Shaft coal-mine, Jefferson County, Ala. [214].
New System for Operating Regenerative Hot-Blast Stoves (WAINWRIGHT) [xlii], 680.
Newton on structure of tin-bearing veins of Black Hills, Dakota [590].
New York: Iron-mining industry for past decade, 745; Product of iron-ore, 721, 725, 727.
New York iron-mine, Marquette County, Mich. [718].
New York, N. Y.: Meeting, February, 1889, proceedings, xxxi.
New Zealand; Magnetic sea-sands [737]; Stamp-mill, driven by electricity, 556, 557.
Niagara Falls: Life-history, 322; Future of, 336; Never at Lewiston, 330 *et seq.*; Rate of recession, 333.
Niagara formation: Near Buffalo, 398 *et seq.*; Near Niagara Falls [325, 327, 328, 331, 335].
Niagara River and geology of region adjacent, 398.
Nichols, E., analysis of coal, 217.
Nickel: *Dakota:* Black Hills, 582; *Kansas:* Logan County, Russell Springs, 636; Analysis, 637; In Ontario, Canada [294, 295, 298].
Nimikon iron-mine, Gogebic Range, Lake Superior [719].
Nitrates and nitrites in impure water, 345.
Nitrogen: Weight of cubic foot of [100]; Specific heat of [100].
Nolan *et al. versus* the Colorado Central Mining Company, features of the case, 796 *et seq.*
Norrie iron-mine, Gogebic Range, Lake Superior, 718.
Norris Geyser Basin, Yellowstone Park, 547, 550, 554.
North Birmingham (see also Birmingham); Blast-furnaces, 61; Visit to [xxii]; Dolomite, 61, 63.
North Carolina: Auriferous sulphides, 313; Pyrites, assay value [314]; Gold product, 314; Manufacture and consumption of phosphoric acid fertilizer, 85.
North Carolina rock, phosphoric acid in, 87.
North Chicago blast-furnace, Chicago, Ill. [756]; Rolling-mill [227].
Northeastern Steel Company, Middlesbrough, England [86].
North Lonsdale blast-furnace, Lancashire, England [756].
Northwestern Colorado Coal-Region (HEWETT) [xxvi], 375.
Norway iron-mine, Menominee County, Mich. [718].
Note on Arsenic Determination (CANBY) [xxi], 77.
Note on Cast-Steel Water-Jackets (TERHUNE) [xxvi], 131.
Note on the Influence of Columbite upon the Tin-Assay (CARPENTER and HEADDEN) [xlii], 633; Discussion by Wm. P. Blake, 785.
Note on the Koepe System of Winding from Shafts (HARDEN) [xlii], 429.

Note on the Nickel-Ore of Russell Springs, Logan County, Kansas (DEWEY) [xxvii], 636.
Note on a Specimen of Gilsonite from Uintah County, Utah (RAYMOND) [xxvi], 113.
Note on the Use of Crude Petroleum as Fuel for Raising Steam at the South Chicago Works (POTTER) [xxxii], 807.
Note upon a Modification of the Reducing Process Used by the Carbon Iron Company (HUNT) [xliii], 678.
Note upon Some Results of the Storage of Water in Arizona (BLAKE) [xliii], 476.
Notes on the Bernice Anthracite Coal-Basin, Sullivan County, Pennsylvania (CLAGHORN) [xlii], 606.
Notes on the Electrolytic Assay of Copper (GLENN) [xlii], 406.
Notes on the Geology and on Some of the Mines of Aspen Mountain, Pitkin County, Colorado (HENRICH) [xxii], 156.
Notes on the Iron-Ores, Fuels, and Improved Blast-Furnace Practice of the Birmingham District (BRAINERD) [xxii], 151.
Notes on the Roasting of the Hudson River Carbonates (OLMSTED) [xxvi], 275.
Notes on the Rosario Mine at San Juancito, Honduras, Central America (LEGGETT) [xxvi], 432.

O'Brien Coal Company, Alabama [218].
Occidental tin-mine, Black Hills, Dakota [595].
Occurrence and Treatment of the Argentiferous Manganese-Ores of Tombstone District, Arizona (GOODALE) [xliii], 767.
Ocotillo gold-mine, San Juancito, Honduras, C. A. [447].
Odanah iron-mine, Gogebic Range, Lake Superior [719].
Odor and color of water, terms describing, 340.
Officers of the Institute; Elected February, 1889, xxxiii; For the year 1888–89, ix.
Ohio: Available tonnage of the coal-fields, 208; Iron-ore product, 724, 725; Iron from lean ores used as "softeners," 254; Silicon-content of Ohio iron, 254; Silicon-irons, 258.
Oil as a Metallurgical Fuel (FELTON) [xxxii], 809.
Oil *versus* coal in the puddling-furnace, 808.
Old Abe shaft, Homestake gold-mine, Black Hills, Dakota [576].
Old Bed iron-mine, Essex County, N. Y. [739, 740, 743, 744].
Old Faithful geyser, Yellowstone Park, 546, 547.
Old Guard silver-mine, Cochise County, Ariz. [775].
Old Shaft coal-mine, Jefferson County, Ala. [214].
Olefiant gas: Value as fuel, 99; Weight of cubic foot of [100].
OLMSTED, INGERSOLL, *Notes on the Roasting of the Hudson River Carbonates* [xxvi], 275.
Onondaga formation: In western New York [250, 325, 327, 398]; In Ontario, Canada [299, 300].
Onondaga salt-wells, Onondaga, N. Y. [110].
Ontario: The geological formations of, 294; The minerals of, 293.
Oolitic limestone, asphalt [374].
Open-hearth blast-furnace, with differential hot-blast stove, 132.
Open-hearth steel practice, improvements in, 678.
Ore-Deposits of the Black Hills of Dakota (CARPENTER) [xlii], 570.
Ore-feeders in use in the Black Hills stamp-mills, 512.
Ore-separator, Wenström magnetic, 599.
Ores of the Black Hills: Gold, 498; The mode of their formation, 573, 574. Ores of Gilpin County, Colorado [541].

Orford Copper Company, New York [408, 410].
Organic remains, on which copper has been deposited [483].
Oriskany sandstone: In Greenbrier County, W. Va. [117], 118, 120; The great ore-bearing formation of Virginia, 119.
Oro Fino gold-mine, Black Hills [586, 587].
Orton, analyses of ferro-silicon and pig-iron, 255.
ORTON, EDWARD, JR., KEEP, W. J., and FLEMING, H. S., *Silicon in Cast-Iron* [xlii], 683.
ORTON, EDWARD, JR., and KEEP, W. J., *Ferro-Silicon, and the Economy of Its Use* [xxvi], 253.
Osmic acid used in examination of impure waters [347].
Otto on phosphate-slag [89].
Ouled Kebbah salt-mine, Ouled Kebbah, Algeria [110].
Owl's Last Hoot silver-mine, Cochise County, Ariz. [775].
Owl's Nest silver-mine, Cochise County, Ariz. [775].
Oxidizing muffle-roasts, losses of silver by volatilization in, 6.
Oxygen, weight of cubic foot of [100].
Oyster shells: Analysis of, 471; Used at Muirkirk Furnace, Md., as a flux, 467.

PAGE, W. N., *Coal Transfer of the Mt. Carbon Company, Limited*, [xlii], 454; *The Glenmore Iron Estate, Greenbrier County, West Virginia* [xxv], 115.
Paint River iron-mine, Menominee Range, Lake Superior [629].
Palæozoic formation: Aspen, Colo. [163, 178, 179, 180]; In the Black Hills [571, 572, 582]; At Tombstone, Ariz. [768].
Palmer Hill iron-mine, Clinton County, N. Y. [747].
Parker, Richard A., on hand-separation of iron-ore at the Champion mine, 729.
Parsons jigs at Bonne-Terre, Mo., 662.
Parsons ore-feeder at Bonne-Terre, Mo., 669.
Parsons-Rittinger Tables, 662, 675.
"Parting Quartzite" at Leadville, Colo. [164].
Patton, Walker County, Ala., coal-mines at, 210, 218, 219.
Pavements: Asphalt, 355, 357, 366 *et seq.*, 371, 375; Preparation of, 361; Extent of in Paris, 362.
PEARCE, RICHARD: Remarks in discussion of Mr. Goodale's paper on *The Occurrence and Treatment of the Argentiferous Manganese Ores of Tombstone District, Arizona*, 774; Method of arsenic determination, 77; First discovered tin in 1877, in the Black Hills, Dakota, 588.
Pechin, E. C.: Remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 146: Resolution proposed at New York meeting, xlii.
Peclet on chimney draught [81].
Pencoat ferro-silicon, 257, 258, 684; Analysis of, 255.
Pennington County, Dakota, gold-ores, 578, 579, 580.
Pennsylvania: Available tonnage of the coal-fields, 208; Bernice coal-basin, 606; Loyalsock-Mahoopany coal-field, 607; Western-Northern anthracite coal-field, 607; Iron: Cornwall ore-banks, 720; Product, 725, 727.
Pennsylvania Steel Works, Steelton, Pa. [150, 227, 809].
Percy, Prof. John, on gold and silver [5, 7], 31.
Perkins, Geo. H., inventor of tin-scrap nail-machine, 496.
Perkins iron-mine, Menominee Range, Lake Superior [718].
Pernolet on grains of solids falling freely in water [640].
Petite Anne Salt-Mine (POMEROY) [xxii], 107.

Petrolene [362].
Petroleum, 357, 358; As blast-furnace fuel [97]; As a preventive of boiler-scale, 353; Value as fuel, 99; In Ontario, Canada, 294, 298, 299; Used as a fuel for raising steam, 807, 808.
PETTEE, W. H., *Biographical Notice of Byron W. Cheever* [xix].
Philadelphia Scale and Testing Works, Pennsylvania [460].
PHILLIPS, WILLIAM B.: *The Chlorination of Low Grade Auriferous Sulphides* [xxvi], 313; *Phosphate-Slag* [xix], 84; On cost of roasting gold-ores at Phœnix mines, North Carolina, 541.
Phœnix gold-mines, Cabarrus County, N. C. [314, 315, 316, 317, 319, 320, 321, 541].
Phœnix Iron Company, Berks County, Pa., magnetic concentration [743].
Phosphate consumption in the United States for fertilizer, 84, 85.
Phosphate of lime in Ontario, Canada [294, 298, 299].
Phosphate-Slag (PHILLIPS) [xix], 84.
Phosphate-slag: Composition of, 89; As a fertilizer, 84, 85, 87, 89.
Phosphatic iron-ores in Southern States [85].
Phosphatic minerals in the Black Hills, Dakota [592].
Phosphoric acid: In Canadian apatite, 87; In Navassa rock, 87; In North Carolina phosphate rock, 87; In South Carolina, Charleston rock, 87; In pig-irons and slags from German works, 92.
Phosphoric fertilizers: A new source of supply, 84 *et seq.*; Available phosphoric acid in, 88.
Phosphorus: Determination of, 750; Determination of, in iron and steel, 100; Distribution of in the Ludington mine, Iron Mountain, Michigan, 616; In the Ludington mine influenced by the presence of jasper, 625, 627; In Alabama iron-ores, 91 [153]; In Georgia iron-ores, 91; In Hudson River carbonate-ores [276]; In iron-ores of Lake Champlain region [721]; In Michigan hematite [752]; In New Jersey iron-ores [722]; In Tennessee iron-ores, 91; In Virginia iron-ores, 122; In iron-ores of Ontario, Canada [299]; In ferro-silicon [256]; In pig-iron from Birmingham, Ala., 96; In steel rails [783]; Amount admissible [425]; Removed from iron-ores by carbon dioxide [629].
Phyllite in Black Hills, Dakota [498, 574].
Pig-iron (see also Iron and Cast-iron): Analyses, 255, 472, 473; Converted to steel by Henderson process, 60 *et seq.*; Cost of, in Sequachee Valley, 45; Grading of pig-iron at Birmingham, Ala., 94; Strength of, 460 *et seq.* [474].
Pig-Iron of Unusual Strength (DEWEY) [xxvi], 460.
Pig-patterns, 427.
Pike iron-mine, St. Lawrence County, N. Y. [747].
Pilot Knob iron-mine, Missouri [723].
Pioneer blast-furnace, Birmingham, Ala. [225].
Pioneer Iron Company [150].
Pioneer Mining and Manufacturing Company, Birmingham, Ala. [143].
Pioneer reduction works, Nevada City, Cal. [3].
Pioneer silver-mine, Aspen, Colo. [178].
Pitkin on electrolytic assay of copper [407].
Pittsburgh salt-wells, Pittsburgh, Pa. [110].
Pittsburgh Testing Laboratory [679].
Pittsburgh and Lake Superior iron-mine, Marquette County, Mich. [718].
Placer-deposits, Black Hills, Dakota, 571.
Platinum ware not suited for use in electrolytic assay of copper [410].
Platt iron-mine, New Jersey [740].

Platt, Franklin, on geology of Bernice coal-basin, 607.
Plattner process applied in the Black Hills, 588.
Plattner on losses of gold and silver in oxidizing-roasting [4, 7, 9].
Plumbago in Ontario, Canada [294].
Pocono sandstone, coal in, 208.
POHLMAN, JULIUS: *Cement Rock and Gypsum Deposits in Buffalo* [xxvi], 250; *The Life History of Niagara* [xxv], 322; Address of welcome at Buffalo [xxv]; On geology of Buffalo, N. Y. [401, 404].
Poisons, metallic, in potable waters, 345.
Polley iron-mine, St. Lawrence County, N. Y. [747].
Polluting ingredients in streams of water, 348.
Polybasite: At Aspen, Colo. [204]; At Rosario mine, Honduras [442].
Poore system of winding from shafts [430].
POMEROY, RICHARD A., *The Petite Anse Salt-Mine* [xxii], 107.
Pomeroy salt-wells, Pomeroy, Ohio [110].
Porphyry: At Aspen, Colo., 175 *et seq.*, 205; Microscopical examination of, 168, 169; In the Black Hills [572, 574, 584, 587]; Affects the richness of the auriferous deposits, 575; In Honduras, Central America [434].
Port Arthur mining district, Ontario, Canada, 296.
Port Henry iron-mine, Essex County, N. Y. [746, 747].
Porter, John B., on iron-ores and coals of Alabama, Georgia, and Tennessee [91, 92].
Porter, J. B. & Co., analysis of coal, 217.
Portland cement, analysis of, 251.
Potsdam formation: In the Black Hills [571, 572, 575, 581, 583]; Ore-bodies in, 585; Carries gold and silver, 586; Does it contain tin? [589].
POTTER, E. C., *Note on the Use of Crude Petroleum as Fuel for Raising Steam at the South Chicago Works* [xxxii], 807.
POTTER, WILLIAM B., *Some Thoughts Relating to the American Institute of Mining Engineers and Its Mission*, 485; *A Present Need in the Engineering Profession*, 380.
Pottsville conglomerate [607, 609].
Pottsville, Pa., thickness of coal, 208.
Pratt coal-bed, Jefferson County, Ala., 152, 209, 210, 211 [223].
Pratt coke, Alabama, 142 [150]; Analysis of, 154; Compared with Connellsville coke, 142.
Pre-glacial history of region adjacent to Niagara Falls, 322 *et seq.*
Pre-glacial valley of Lake Erie, 322 *et seq.*; of Lake Ontario [324].
Present Need in the Engineering Profession (POTTER), 380.
Present Status of Electrical Transmission of Power (ROTHWELL) [xxv], 555.
Pribram, Bohemia, magnetic ore-separator [736].
Price, Thos.: Experiments in purifying alloys of gold [32]; On the Miller process of refining gold [32].
Priming of water in steam-boilers, cause and cure, 351.
Proceedings of Meetings: Birmingham, xix; Buffalo, xxiv; New York, xxxi.
Producer-gas, value as fuel, 99.
Product of Bernice anthracite coal-basin, 614.
Prominent Sources of Iron-Ore Supply (BIRKINBINE) [xliii], 715.
Prompter silver-mine, Cochise County, Ariz. [775].
Proposed Rail-Sections (HUNT) [xxxii], 778.
Providence stamp-mill, Nevada City, Cal. [504].

Psilomelane at Butte, Mont. [774].
Pterygotus in Buffalo cement-rock, 251.
Publications of the Institute, xi.
Puddling-furnace fuel, coal *versus* oil, 808.
Pumps: Heald & Cisco's centrifugal pump for sand and water [668].
Puritan iron-mine, Gogebic Range, Lake Superior [719].
Putnam County, N. Y.: Iron-ore concentration, 730; Tilly Foster iron-mine, 758 *et seq.*
Pyrite: In the Black Hills, Dakota [582]; Loss in chloridizing-roasting, 15, 16.
Pyrolusite: At Butte, Mont. [774]; At Rosario mine, Honduras [442].
Pyromorphite at Rosario mine, Honduras [442].
Pyrrhotite: In the Black Hills [574, 581, 582]; In Ontario, Canada [294].

Quartzite at Aspen, Colo. [163].
Quaternary placers in the Black Hills [571].
Quicksilver (see Mercury and Amalgamation-works).

Ragland, St. Clair County, Ala., coal-mines at, 210, 217.
Rails: Analyses, 234, 237; Affected unfavorably by hot rolling, 246; Effect of rapid rolling on structure [423]; Inspection, 240, 244; Relation of chemical composition to wear, 233, 237; Tests [235], 239, 244, 247; Wear affected by shape of section, 421 *et seq.*
Rail-Sections (DELANO) [xxxii], 421.
Rail-sections, 226 *et seq.*, 230, 232 *et seq.*, 243. 778; Dudley's New York Central section, 783.
Rails, steel, 226 *et seq.*; Specifications for, 238; Developments in steel-rail manufacture, 226.
Rainfall in northwestern Colorado, 376.
Rammelsberg deposit and Black Hills deposit compared, 575.
Rankine on the steam engine [80, 81].
Rapid Method for the Reduction of Ferric Sulphate in Volumetric Analysis (JONES) [xlii], 411; Discussion by J. K. Eveleth, 757.
Rapid Method for the Determination of Phosphorus in Certain Ores (WOODBRIDGE) [xliii], 750, 753.
Rattlesnake silver-mine, Cochise County, Ariz. 774].
Raymond and Campbell hot-blast stove [463].
RAYMOND, R. W.: *End-Lines and Side-Lines in the U. S. Mining Law* [xxxi], 787; *Note on a Specimen of Gilsonite from Uintah County, Utah* [xxvi], 113; *Soaping Geysers* [xxv], 449; Remarks in discussion of Mr. Gordon's paper on *Large Furnaces on Alabama Material*, 148; On crushing-weight of burden in blast-furnace, 148; On Leadville ore-deposits, 449; On the law of the apex, 796; On lode locations [790, 796]; Glossary of mining and metallurgical terms [xi, xii].
Recent placers in the Black Hills [571].
Red Mountain, Alabama, fossil-ores, 152.
Red Mountain, Aspen, Colo. [180].
Red Mountain district, San Juan, Colo.: Geology, 262; Hot-spring formations, 261.
Red oxide of copper in the Black Hills [581].
Reducing process used by the Carbon Iron Company, 678.
Reduction of ore in blast-furnace as affected by the velocity and the tension of the gases, 282.

Reduction-works (see also Chlorination-works, Concentration works, Gold-mills, Lixiviation-works, Smelting-works, and Stamp-mills): *California:* Nevada County; Nevada City, Pioneer [3].
Refining alloys of gold: Aiken's process for, 30, 31; Miller's process for, 30 *et seq.;* Thompson's process for, 30, 31.
Regenerative furnace, Siemens's [134].
Regenerative hot-blast stove, 132, 680.
Reopening of the Tilly Foster Iron Mine (McDowell) [xliii], 758.
Report of Council for the year ending February, 1889, xxxiii.
Republic iron-mine, Marquette County, Mich. [717]; Concentration-works at [728].
"Retarded coke," an artificial fuel, 678.
Retsof salt-mine, Piffard, N. Y., visit to [xxx].
Review of the Iron-Mining Industry of New York for the Past Decade (Smock) [xliii], 745.
Rhein Steel-works, Ruhrort, Germany [93].
Rhyolite in San Juancito, Honduras, C. A. [438, 447].
Ridsdale and Stead, average composition of English slag, 86.
Rittinger's formula for bodies falling freely in a liquid [639].
Rittinger-tables with Parsons's improvement, 662, 675.
Roasting-furnace at Phœnix gold-mine, North Carolina, 317, 318.
Roasting gold-ores: Losses in, 3; In North Carolina, cost of [541].
Roasting Hudson River carbonates, 275.
Roasting of Murchie pyrite, 6.
Roasting-kilns for carbonate ores [275].
Robert Emmet silver-mine, Aspen, Colo. [170, 176].
Roberts, Frank C., *Calculations of the Available Heat and the Required Dimensions of Chimneys, Combustion-chambers and Gas-Burners in the Use of Blast-Furnace Gases for Firing Boilers* [xxi], 78.
Robertson, Kenneth, *The Grading of Birmingham Pig-Iron* [xxii], 94.
Robinson, Theo. W., *The Effect of Velocity and Tension of Gases on the Reduction of Ores in the Blast-Furnace* [xxi], 282.
Rock-breakers in use at Homestake Mill, Lead City, Dak., 509.
Rogers, E. M., *The Equalization of Load on Winding-Engines by the Employment of Spiral Drums* [xxvi], 305.
Rolker, C. M., on geology of Rosario mine, San Juancito, Honduras, C. A. [432, 435].
Rolling-mill, Fritz [228].
Rolling-mills: *New York:* Rensselaer County; Troy [235]; *Ohio:* Cuyahoga County; Cleveland, Cleveland Rolling Mill Company [150]; *Pennsylvania:* Cambria County; Johnstown, Cambria [227]; Dauphin County; Steelton, Pennsylvania Steel Works [227].
Rosario gold-mine, San Juancito, Honduras, C. A., 432 *et seq.*
Rothwell, R. P., *Present Status of Electric Transmission of Power* [xxv], 555.
Rowan County, N. C., Gold Hill gold-mine [314].
Rules of the Institute, xiii; Amendments to, xxvii, xli.
Russell lixiviation process [45].
Russell Springs, Kansas, nickel-ore, 66.
Russian auger used at Petite Anse salt-mine [112].
Rutile in the Black Hills, Dakota [786].
Ruttmann, F. S.: On concentration of iron-ore at Lyon Mountain, N. Y. [732]; On the Tilly Foster ore-body [759].

Saginaw salt-wells, Saginaw, Mich. [110].
Salisbury Iron Company, strength of iron, 463.
Salisbury iron-mine, Marquette County, Mich. [718].
Salina formation in western New York [398 *et seq.*].
Salom and Westesson, analysis of Bernice anthracite, 610, 615.
Salt in Ontario, Canada [294, 298, 300].
Salt, increase of, increases loss of gold and silver in chloridizing-roasting, 14, 17.
Salt-mines: UNITED STATES: *Louisiana:* Iberia Parish, Petite Anse Island, Avery mine, 107 *et seq.; New York:* Livingston County, Piffard, Retsof [xxx]. OTHER COUNTRIES: *Algeria:* Ouled Kebbah [110]; Jebel-Melah [110]; *Austria:* Hallstadt [110]; Wieliczka [110]; *Bavaria:* Berchtesgaden [110]; *German Lorraine:* Vic [110]; *Germany:* Stassfurt [110]; *Ireland:* Carrick-fergus [110]; *Santo Domingo* [110]; *Tyrol:* Hall [110].
Salt-mining and preparation at Petite Anse Island, Louisana, 107 *et seq.*
Salt-quarry (see also Salt-mines): *Siberia:* Iletsk [111].
Salt-wells: UNITED STATES: *Michigan:* Saginaw [110]; *New York:* Onondaga [110]; *Ohio:* Hocking Valley [110]; Pomeroy [110]; *Pennsylvania:* Pittsburgh [110]; *West Virginia:* Kanawha [110]. OTHER COUNTRIES: *Canada:* Ontario, Goderich [110]; *England:* Droitwich [110]; Cheshire [110]; *German Lorraine:* Dieuze [110].
Sampling and Testing Works, St. Louis, Mo., 387.
Sampling gold-ores in the Black Hills, methods of, 499.
Samson iron-mine, Humboldt, Mich. [734].
San Jacinto claim, Aspen Colo. [178].
San Juan mining district, Colorado, hot-springs of, 262.
San Juancito, Honduras, Rosario gold-mine, 432 *et seq.*
Sandberg, C. P.: Discussion on steel-rails and specifications, 242; On rail specifications, 244; On standard rail-sections, 243.
Sandberger, F., on vein-formation [448].
Sandstone in Ontario, Canada [294].
Sanitary analysis of water, 341, 343.
Sanitary condition of mines and metallurgical works [339].
Santo Domingo salt-mine, Santo Domingo [110].
Sawatch Range, Colo. [161, 177, 180].
Scale, boiler: The cause of, 350, 351; Preventative, 351, 353, 354.
Schaeffer, Prof. C. A.: Analysis of tantalite, 593; On the columbite (tantalite) of the Black Hills, 633.
Schaufuss, Erich C., biographical notice of, 419.
Scheibler's process for concentrating phosphoric acid in phosphate-slag [94].
Schiller silver-mine, Aspen, Colo. [168, 170, 176, 177].
Scotch pig-iron as a softener for other irons, 684.
Segregated Old Abe gold-deposit, Black Hills [576].
Septic poisons in water, 344, 346.
Sequachee Valley: Cost of coke in, 48; Cost of pig-iron in, 45, 49.
Serpentine in Ontario, Canada [294, 298].
Setterwall, analysis of pig-iron, 255.
Sewage, ammonia in, 344.
Seyssel asphalt, France [356], 360 [373].
Shamokin anthracite district, Pennsylvania [611].
Shelby County, Ala.: Coal-mines at, 210, 221; Coal product in 1887, 207.
SHIMER, PORTER W., *Determination of Phosphorus in Iron and Steel* [xxi], 100.

Shrinkage in cast-iron explained, 697.
Shirtliff iron-mine, Jefferson County, N. Y. [747].
Sicily, bituminous limestones of, 362.
Side-lines and end-lines in U. S. mining-law, 787.
Siemens's regenerative furnace, 134.
Silicon in Cast-Iron (KEEP, FLEMING and ORTON) [xlii], 683.
Silicon: In pig-iron affected by rapid driving, 144, 151; In pig-iron from Ohio, 254; In Southern irons, 95; In steel-rails [783]; Its influence on the chilling of iron, 705 *et seq.*; On the deflection, 695; The fluidity, 708 *et seq.*; The grain, 710; The hardness, 702 *et seq.*; The shrinkage, 697; The strength, 689 *et seq.*; Expels carbon from cast-iron, 684; Its effect on the casting of steel-ingots, 245; In pig-iron, and in ferro-silicon, affected by remelting the metal, 256 *et seq.*; As a softener for foundry-iron, 253 *et seq.*; Reactions with alkaline carbonates, 542; Determination of, in ferro-silicon, 542; Graphitoidal, 542; In aluminum, 542, 544.
SILLIMAN, J. M., *A Water-Manometer and Anemometer* [xxii], 66.
Silurian formation: At Aspen, Colo. [157, 163, 164, 173, 178, 179, 185]; Near Buffalo, N. Y., 250 [398]; In northwestern Colorado [376]; In West Virginia [116 *et seq.*]; In Ontario, Canada [294]; Not seen in the Black Hills [571].
Silver: In the Black Hills [571]; In Ontario, Canada [294], 296 [298, 300]; At the Rosario mine, Honduras [442]; Effect of chlorine on, at cherry-red heat, 34; Loss in chloridizing-roasting, 13, 18; Loss in chloridizing-roasting increases with amount of salt used, 14; Loss in chloridizing-roasting, effect of increased temperature on, 19; Loss in chloridizing-roasting, effect of time on, 15; Losses in oxidizing-roasting, 4, 5, 6.
Silver Bell silver-mine, Cochise County, Ariz. [775].
Silver Islet silver-mine, Ontario, Canada [296].
Silver-mills (see Amalgamation-works, Chlorination-works, Concentration-works, Gold-mills, Lixiviation-works, Reduction-works, Smelting-works, and Stamp-mills).
Silver-mines: *Arizona:* Cochise County; Big Find [775]; Blue Monday [775]; Bunker Hill [775]; Burchard [775]; Contact [775]; Fourth of July [775]; Hard Up [775]; Independence [775]; Knoxville, 767 *et seq.*; Longfellow [775]; Luck Now [775]; Luck Sure, 767, 774 [775]; Lucky Cuss, 767, 774 [775]; Mammoth [775]; McCann [775]; Old Guard [775]; Owl's Last Hoot [775]; Owl's Nest [775]; Prompter [775]; Rattlesnake [775]; Silver Bell [775]; Sunset [775]; Triangle [775]; Ural [775]; Wedge, 767 [775]; *Colorado:* Chaffee County; St. Elmo, Mary Murphy [159]; Ouray County; Congress [264]; Guston [264]; Yankee Girl [264]; Pitkin County; Aspen, Aspen, 161, 170, 173, 193 *et seq.*; Aspen-Mammoth [170]; Bonnybel [170 *et seq.*, 202]; Chloride [159, 170, 202]; Connamara [170, 173, 200 *et seq.*]; Durant [170]; Emma [160, 170, 191 *et seq.*]; Robert Emmett [170]; Forrest [170, 175]; Franklin [170]; Galena [178]; Golconda [170, 176]; Jessie [170]; La Salle [170, 173]; Last Scheme [167]; Late Acquisition [178]; Little Giant [170, 171, 176]; Millinee [170]; Missouri Boy [178]; Monarch [178]; Pioneer [178]; Schiller [170, 177]; Silver Star [170 *et seq.*, 202]; Smuggler [159]; Spar [159, 170], 181 *et seq.*; Stillwell [170]; Traynor [178]; Vallejo [170, 189, 190]; Visino [173, 174, 197 *et seq.*]; Washington [170, 186 *et seq.*]; *Dakota:* Black Hills; Iron Hill, 584.
Silver-ores, analyses of, 769.
Silver-lead ores: *Dakota:* Black Hills, 582.

Silversmith-case, involving validity of end-lines in a mining claim, 787, 794.
Silver Star mine, Aspen, Colo. [170, 202].
Silver volatile as oxide and as chloride [18].
Size of gold particles in auriferous pyrites [29].
Sizing in the concentration of iron-ores, 733.
Sizing of ores for jigging: Is close sizing advantageous? 637 *et seq.*
Slag: Analysis of Muirkirk Furnace slag, 470; In Birmingham district, Ala. [140]; Phosphate-slag as a fertilizer, 84 *et seq.*
Slagging-valve for tuyeres, 389.
Slimes, their treatment in concentration-works, 656.
Sloss blast-furnace, Birmingham, Ala. [152].
Sloss Steel and Iron Company, North Birmingham, Jefferson County, Ala., 61, 211, 212.
Smelting by electricity, 559.
Smelting-works (see also Chlorination-works, Concentration-works, Gold-mills, Lixiviation-works, Reduction-works, and Stamp-mills): *Arizona:* Cochise County; Tombstone, 771, 773.
Smith, E. A., on Cahaba coal-field, Alabama, 209.
Smith, J. C., analysis of Low Moor flue-dust, 130.
SMITH, OBERLIN, *Nails from Tin-Scraps* [xliii], 495.
SMOCK, JOHN C., *A Review of the Iron-Mining Industry of New York for the Past Decade* [xliii], 745.
Smuggler Mountain, Aspen, Pitkin County, Colo. [156, 159, 180].
Smuggler silver-mine, Aspen, Colo. [156, 159].
Sneyd colliery, North Staffordshire, England [431, 432].
Soaping Geysers (HAGUE) [xliii], 546.
Soaping Geysers (RAYMOND) [xxv], 449.
Soap-test for hard water, 354.
Soft-ore bodies, mining in, at Low Moor, 103.
"Softeners" of pig-iron [684, 702, note].
Some Thoughts Relating to the American Institute of Mining Engineers and its Mission (POTTER), 485.
South Chicago steel-works, use of petroleum as fuel at, 807.
South Carolina, manufacture and consumption of phosphoric acid fertilizer, 85.
Southern iron-ores: Irregular in composition, 144; Readily reducible, 147.
Spain: Bilbao iron district [719]; Bituminous limestones [362].
Spanish pyrites, influence in cheapening sulphuric acid [84].
Spar silver-mine, Aspen, Pitkin County, Colo. [159, 170, 176], 181 *et seq.*
Sparre on grains of solids falling freely in water [640].
Specifications for manufacture of steel-rails, 226, 238.
Specifications for structural material, importance of standards for [493].
Spencer, J. W., on geology of Niagara Falls [322].
Spense, automatic desulphurizing furnace [541].
Spilsbury, E. G., on treatment of southern gold-ores [315, 320].
Spiral-drums to equalize load on winding-engines, 305.
Spiral Weld Tube Works, East Orange, N. J., visit to [xliii].
Spitzkasten at St. Joseph Lead Works, Bonne-Terre, Mo. [662], 673.
Spodumene, large crystals in the Black Hills, Dakota, 591.
Sprague electric motors [558, 563].
Spuyten Duyvil rolling-mills, New York [227].
Stamp-mills (see also Chlorination-works, Concentration-works, Gold-mills, Lix-

iviation-works, Reduction-works, and Smelting-works): *Arizona:* Cochise County; Tombstone [771, 773]; *California:* Nevada County; Nevada City, Providence [504]; *Dakota:* Lawrence County; Central City, Cassel, 500; Central City, Father de Smet [498], 500 *et seq.*; Lead City, Golden Star, 500 *et seq.*; Highland, 500 *et seq.*; Homestake, 500 *et seq.*; Terraville, Caledonia, 500 *et seq.*; Deadwood, 500 *et seq.*; Golden Terra, 500 *et seq.*

Stamp-mills in the Black Hills, description and general features of, 501 *et seq.*

Standard specifications for structural material [493].

Stassfurt salt-mine, Stassfurt, Germany [110].

Staten Island, N. Y., iron-district (limonites) [745, 749].

Statistics of the Alabama coal-fields for 1887, 206.

Status of Associates changed: At Birmingham, xxi; at Buffalo, xxix; at New York, xli.

St. Clair County, Ala.: Coal-mines of, 210, 217; Coal product in 1887, 207, 210.

Stead and Ridsdale: Crystals in basic slag [89]; On average composition of English slag, 86.

Steam-boilers, using the waste gas of a blast-furnace as fuel for, 50.

Steam-engine, Rankine on [80, 81].

Steam, specific heat [100].

Steam-stamps in use in the Black Hills [528].

Steel (see also Bessemer steel): Analyses, 64; Use of phosphatic ores in making, 90; Determination of phosphorus in, 100; From pig-iron by Henderson process, 60 *et seq.*

Steel rails (see also Rails): Analyses, 234, 237; The development of, 226 *et seq.*; Dudley sections, 783; Papers by Messrs. Dudley, Holley, and Sandberg [xii]; Proposed sections, 778.

Steel Rails and Specifications for their Manufacture (HUNT) [xxv], 226.

Steel rails, specifications for, 238, 425.

Steel water-jackets, cast, 131.

Steel-works: *Alabama:* Jefferson County; North Birmingham, Henderson Steel and Manufacturing Company, 60 *et seq.*; Sloss Steel and Iron Company, 61; *Pennsylvania:* Allegheny County; Pittsburgh, Carbon Iron Company [678]; Cambria County; Johnstown, Cambria [227]; Chester, Chester Steel Castings Company, 131; Dauphin County; Steelton, Pennsylvania Steel Company [150, 227, 809]; *Germany:* Ruhrort, Rhein Steel Works [93]; *England:* Middlesbrough, Northeastern Steel Company [86].

Stephanite at Aspen, Colo. [204].

Sterling iron-mine, Rockland County, N. Y. [746].

Stetefeldt, C. A., on losses of gold in roasting, 10, 11, 36.

Stetefeldt furnace, loss of gold in roasting in [11].

Stevens Institute of Technology, Hoboken, N. J.; Session at [xliii].

Stickney, George H., analysis of Baltimore ores, 471.

Stickney Iron Works, Baltimore, Md., 471; Strength of Stickney iron, 463.

Stillwell silver-mine, Aspen, Colo. [170].

St. Joseph Lead Works, Bonne-Terre, Mo., 564, 637 [644, 650, 654, 659].

St. Lawrence County, N. Y., iron-district [745], 747.

St. Louis Sampling and Testing Works, 387.

Storage of water in Arizona, 476.

Stoves for hot-blast, regenerative, 680.

Stream-tin: Analyses, 596; In Black Hills, Dakota, 596.

Structural steel, analysis of, 679.

Sub-carboniferous age; Greenbrier limestones, West Virginia [117], 119, 121; Vespertine shale, West Virginia [117], 119, 121, 122.
Sudbury copper-mines, Ontario, Canada [295].
Suisse on Koepe system of winding [432].
Sullivan County, Pa, Bernice coal-basin, 606.
Sulphur in steel rails [783].
Sulphur, its effect in chloridizing-roasting, 776.
Sulphurous acid to condense gold from chlorine gas, 37, 38.
Sunday Lake iron-mine, Gogebic Range, Lake Superior [719].
Sunset silver-mine, Cochise County, Ariz. [775].
Superior iron-mine, Gogebic Range, Lake Superior [719].
Super-phosphate compared with phosphate-slag as a fertilizer, 89.
Supplementary Note on Blast-Furnace Lines (WALSH) [xliii], 754.
Swank, James M., estimate of product of iron-mines of the U. S., 749.
Sylvanite in Ontario, Canada [294, 298].

Tables in stamp-mills, Parsons and Rittinger, 675.
Taliaferro, E. T., address of welcome at Birmingham [xix].
Tantalite: Analysis, 593; In the Black Hills, Dakota, 592, 593, 633 [786]; Associated with tin, 592.
Tar-springs, 358.
Tate iron-mine, St. Lawrence County, N. Y. [747].
Teaser silver-mine, Aspen, Colo. [171, 173, 176].
Technical education [381]; Papers and discussions on [xii].
Tecumseh Iron Company, strength of iron, 463.
Tellurides supposed to be present in Murchie mine [3, 6].
Tellurides of gold and silver, losses in roasting [5, 9].
Tellurium in ores at Tombstone, Ariz. [771].
Temperature of combustion of blast-furnace gases, 79; Of geyser waters, Yellowstone Park [554].
Tennessee Coal, Iron and Railway Company, Ensley City, Ala., 135; Works visited [xxii].
Tennessee: Analyses of coals, 47; Coal-mines, 47; Cost of labor in Sequachee Valley, 49; Iron-ore product, 723, 725; Manufacture and consumption of phosphoric acid fertilizer, 85; Tar-springs [358].
TERHUNE, RICHARD H., *Note on Cast-Steel Water-Jackets* [xxvi], 131.
Terra gold-mine, Black Hills, Dakota, 573, 574, 575, 577.
Tertiary formation; In the Black Hills [571, 589]; In northwestern Colorado [377].
Test for metals in potable water, 346.
Testing and sampling works, St. Louis, Mo., 387.
Testing laboratory at Pittsburgh, Pa. [679].
Testing of engineering materials, importance of making provision for, 386, 493.
Testing-machines: Gill's, 461; Philadelphia Scale and Testing Works, 460; United States testing-machine at Watertown Arsenal, Mass. [386, 461].
Tetrabasic phosphate, 81, 84.
Tetrahedrite in Ontario, Canada [294].
Texas: Deposition of copper on fossil plants [483]; Glance-pitch, 359.
Theal iron-mine, Putnam County, N. Y. [746].
Thenardite in Arizona [480].
Theory of formation of ore-deposits at Aspen, Colo., 204.
Theory of jigging, 638.

Thies, Adolph, manager of Phœnix gold-mine, North Carolina, 316.
Thies chlorination process, 316.
Thomas coke-ovens, 212.
Thomas Iron Company's blast-furnace, Alabama, visited [xxii].
Thomas Iron Company, Hokendauqua, Pa. [461].
Thomas process for dephosphorizing iron [86, 92].
Thomas slag, 81, 84.
Thomassy on geology of Petite Anse Island [108].
Thompson, Lewis: Method of assaying and purifying gold, 7 [31]; Process for refining alloys of gold, 30, 31.
Tilly Foster iron-mine, Putnam County, N. Y. [746]; Reopening of, 758 *et seq.*
Timber-consumption in mines, circular of inquiry, 266.
Timber in Greenbrier County, W. Va., 121.
Timbering: Best woods for mine timbering, 269; Consumption of wood in mine-timbering, 265 *et seq*
Timbering in soft ore-body at Low Moor, Va., 104 *et seq.*
Tin: In the Black Hills, 588; Yield of, 596; Influence of columbite upon the tin assay, 633.
Tin-mines: *Dakota:* Black Hills [571], 595; Etta, 591, 592; Cow Boy [786]; Custer County; Tin Mountain, 591.
Tin-ore: Analysis, 595; In the Black Hills, 595 *et seq.*
Tin-ore in the Black Hills not found in true granite, 591, 592.
Tin-ore of the Black Hills, concentration by jigging, 597.
Tin Reef tin-vein, Black Hills [590].
Tin-scrap made into nails, 495.
Tipples, coal, on the Kanawha River, 455.
Titanium in iron-ores of Ontario, Canada [299].
Todd iron-mine, Putnam County, N. Y. [746].
Tombstone district, Arizona: Argentiferous manganese-ore, 767.
Tombstone Mining and Milling Company, Tombstone, Ariz. [771, 773].
Tonawanda, N. Y., natural gas, 403.
Topography of Glenmore Iron Estate, Greenbrier County, West Virginia, 116.
Tourmaline associated with tin in the Black Hills [591, 595].
Trachyte in Honduras, Central America [434].
Tracy City coal-mine, Tennessee [209, 211].
Transactions of the Institute: Prices of the volumes [xii]; Stock of volumes, xxxv, xxxvi.
Transmission of power by electricity, 555.
Transpiration of gases in blast-furnace, 282.
Trasenster, M. L., on Koepe system of winding [432].
Traynor silver-mine, Aspen, Colo. [178].
Treatment of argentiferous manganese-ores of Tombstone, Ariz., 767.
Treibach blast-furnace, Treibach, Austria [756, 757].
Triangle silver-mine, Cochise County, Ariz. [775].
Trenton formation near Niagara River [402].
Triassic formation: At Aspen, Colo. [156, 157, 180]; In the Black Hills [571]; In New Jersey, containing copper [483]; In Honduras, Central America, [433, 435, 438].
Trinidad asphalt, 361, 362, 368.
Triphylite in the Black Hills, Dakota [592.]
Triunfo gold-mine, Honduras, Central America [447].

Troy Steel and Iron Company, Troy, N. Y. [235].
Trussville blast-furnace, Ala., visit to [xxiii].
Tullock automatic ore-feeder, 512.
Tully limestone in western New York [400].
Turner, Prof. Thomas, on the effect of silicon upon cast-iron, 253, 684 *et seq.*
Turtelotte Park, Aspen, Colo. [158, 171, 175, 178].
Tuscaloosa County, Ala.: Coal-mines of, 210, 221; Coal product in 1887, 207, 210.
Tuyere Slagging-Valve (COOK) [xxvi], 389.

Uintahite from Uintah County, Utah, 113.
Union blast-furnace, Chicago, Ill. [756, 757].
Union Bridge Works, Buffalo, N. Y., visited [xxix].
Union Steel Company, Chicago, Ill. [243].
United States Engineer Post and Torpedo Station, Willet's Point, L. I., visited [xliv.].
United States geological map, by C. H. Hitchcock [xii].
United States, product of iron, 715.
Upper Geyser Basin, Yellowstone Park, 546, 554.
Ural silver-mine, Cochise County, Ariz. [775].
Uranium associated with tin in the Black Hills [593].
Utah, Uintah County: Gilsonite, 113 [359]; Glance-pitch [359].
Utica formation in western New York [402].

Val de Travers, France, asphalt-mine [360].
Vallejo silver-mine, Aspen, Colo. [170, 176, 189, 190, 196].
Valves: Tuyere slagging-valve, 389.
Varnish made from asphalt, 364.
Vein-stones of the tin-lodes of the Black Hills, 594.
Vein-structure of tin-bearing veins, Black Hills, Dakota, 590.
Velocity of spheres falling in narrow tubes, 643, 657.
Venezuela, tar-springs [358].
Vermilion iron-range, Minnesota, 716 *et seq.* [753]; Product of iron-ore, 716, 717, 725, 727.
Vespertine formation in Greenbrier County, W. Va. [117, 119, 121, 122].
Veteran Tunnel, Aspen, Colo. [171, 176, 190].
Vic salt-mine, Vic, German Lorraine [110].
Vincent on structure of the tin-bearing veins of the Black Hills, Dakota, [590].
Virginia: Iron-ore product, 723, 725; Manufacture and consumption of phosphoric acid fertilizer, 85.
Virginia and Alabama Mining and Manufacturing Company, Alabama [218].
Visino Incline and Tunnel, Aspen, Colo. [171, 173 *et seq.*, 191, 197 *et seq.*, 205].
Volatility of gold, 3 *et seq.*; In atmosphere of chlorine-gas, 19, 25 *et seq.*
Volatilization of silver, losses by, in roasting gold-ores, 3 [6].
Volumetric analysis, rapid method for reduction of ferric sulphate, 411.
Vorce on the influence of aluminum on cast-iron [473].
Voudy, analysis of iron-ore, 744.
Vulcan iron-mine, Menominee County, Mich. [718].

Wages: In Black Hills stamp-mills, 532; In Sweden, 604; Of coal-miners in Alabama, 222.
Wagner on phosphate-slag [89].

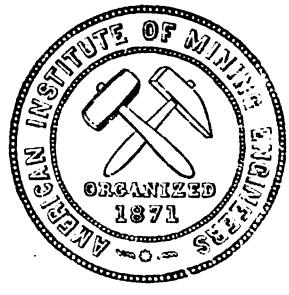
Wagoner, Luther, on falling-velocity of grains in water and glycerine, 653.
WAINWRIGHT, JACOB T.: *A Differential Regenerative Hot-Blast Stove and Its Application to an Open-Hearth Blast-Furnace* [xxvi], 132; *The Feasibility of Using Cheaper Fuels in the Blast-Furnace* [xxii], 96; *A New System for Operating Regenerative Hot-Blast Stoves* [xlii], 680.
Walker County, Ala.; Coal-mines of, 210, 218, 220; Coal product in 1887, 207, 210.
Walnut Grove Dam, Yavapai County, Ariz. [476].
Walrand on phosphorus in pig-iron destined for basic process [92].
WALSH, EDWARD, JR.: *Supplementary Note on Blast-Furnace Lines* [xliii], 754; On blast-furnace process, 282.
Wanklyn on copper, iron, and lead in potable water [346].
WARREN, WILLIAM Y., *Note on the Cultivation of Mushrooms in Abandoned Mines at Akron, N. Y.* [xxv], 248.
Warrior coal-field, Alabama [148], 207 *et seq.*
Warrior, Jefferson County, Ala., coal-mines at, 210, 212 *et seq.*
Warwick blast-furnace, Pottstown, Pa.: Tuyere slagging-valve in use at, 389; Use of anthracite and coke, separate and mixed, 124.
Washing Alabama coal to improve for coking, 141, 145.
Washington, D. C., asphalt pavements in, 367.
Washington silver-mine, Aspen, Colo. [170, 176, 186 *et seq.*].
Washoe process for gold- and silver-ores in the Black Hills, 588.
Wassaic blast-furnace, Amenia, Dutchess County, N. Y., 469, 472.
Waste gas from blast-furnace used as fuel for steam, 50.
Water: Amount of solid residue on evaporation, 348; Analysis for sanitary purposes, 341; Analyses, 353; When unfit for making steam, 350; Falling-velocity of grains in, 653, 657; Impurities of, 338; Its specific heat [100].
Water-Gas as a Steam-Boiler Fuel (JACOBUS) [xxvi], 300.
Water-gas: Calorific power of, 302; Value as fuel, 99; Analyses of, 300, 301.
Water-jackets of cast-steel, 131.
Water-lime formation in western New York [250, 252, 399].
Water-Manometer and Anemometer (SILLIMAN) [xxii], 66.
Water-storage in Arizona, 476.
Water-tube steam-boiler, using the waste-gas of a blast-furnace as fuel, 50.
Watertown arsenal, testing machine at [386, 461].
Watts Coal and Iron Company, Alabama [141].
Watts coke, analysis of, 154.
Wayne County, N. Y., iron district [745], 748.
Weber shales at Aspen, Colo. [166, 176 *et seq.*, 187, 189, 190, 193, 196, 197, 199].
Webb, James F., address of welcome at Birmingham [xix].
Weekes, Edward F., on dumping-cradle at the Chapin mine, 566.
Wedding on phosphate-slag [89].
Wedge silver-mine, Cochise County, Ariz., 767, 775.
Weed iron-mine, Columbia County, N. Y. [748].
Welding by electricity, 559.
Welding of gold buttons at low temperature [30, note].
"Wellston" pig-iron: Analyses, 255; Wellston white iron [701].
Wendt, A. F.: On ores of Copper Basin, Ariz. [483]; On the iron-ores of Putnam County, N. Y. [759].
Wenström Magnetic Separator (COOK) [xlii], 599.
Wenström magnetic ore-separator, 738, 739.
West Aspen Mountain, Colorado, geological section of, 180.

Western-Northern anthracite coal-field, Pennsylvania, 607.
Westesson and Salom, analysis of Bernice anthracite, 610, 615.
West Virginia: Available tonnage of the coal-fields, 208; Coal in Greenbrier County, 119, 121; Limestones in Greenbrier County, 120, 121; Manufacture and consumption of phosphoric acid fertilizer, 85; Timber in Greenbrier County, 121; Coke, sustaining power of, 147.
Whirlpool, Niagara Falls [328].
White and Howell furnace [771].
Whitney, A. W., analysis of Muirkirk pig-iron, 471.
Whittemore, D. J., on rail-sections, 424.
Whitwell hot-blast stove [53, 59].
Whitwell coal-mine, Tennessee [209, 211].
Wieliczka salt-mine, Wieliczka, Austria [110].
Wigglesworth, Thos. H., on gilsonite from Uintah County, Utah, 113 *et seq.*
Wilcox, McKean County, Pa., natural gas at [405].
Wiley on phosphate-slag [89].
WILLIAMS, HENRY J., *The Determination of Silicon in Ferro-Silicons; Its Occurrence in Aluminum as Graphitoidal Silicon, and a Study of Its Reactions with Alkaline Carbonates* [xxxii], 542.
Williamson blast-furnaces, Birmingham, Ala. [212].
Wimmer, on the Rammelsberg ore-deposit, 576.
Winding-engines, equalizing the load by spiral-drums, 305.
Winding from shafts by Koepe system, 429.
Winifrede Coal Company, West Virginia [455].
Winkler on loss of gold in roasting metallic sulphides [5].
Winthrop iron-mine, Marquette County, Mich., 718.
Wire-rope tramway at Tilly Foster iron-mine, Putnam County, N. Y., 761 *et seq.*
Witherbee, T. F., on experiment in substituting anthracite on top of charcoal, at Fletcherville furnace [146].
Wolf Den Hollow coal-mine, Jefferson County, Ala., 214.
Wolfram in the Black Hills, Dakota [786].
Wood's ore-bank, near Muirkirk furnace [465].
WOODBRIDGE, T. REED: *A Rapid Method for the Determination of Phosphorus in Certain Ores* [xliii], 750, 753; Analysis of iron-ore, 721, 744.
Woods best suited for mine-timbering, 269.
Woodstock Coal and Iron Company, Alabama [225]; Works visited [xxiii].
Woodward, Jefferson County, Ala., coal-mines at, 210, 215.
Woodward Iron Company, Woodward, Jefferson County, Ala., 153; Experiments in washing coal [141].
Woodward, R. S., survey of Niagara Falls, 333.
Wrbna blast-furnace, Eisenerz, Austria [756].
Wrightson on phosphate-slag [89].
Wulfenite at the Rosario mine, Honduras [442].
Wyatt, Dr. Francis, on construction of condensing towers, 41.
Wyatt on phosphate-slag [89].
Wyoming: Black Hills, tin, 589.
Wyoming County, Pa., coal in, 607, 609.

Yale and Towne Company, Stamford, Conn. [462].
Yankee Girl mine, Red Mountain district, Colorado [264].
Yavapai County, Ariz., Copper-deposits of Copper Basin, 479.

Yield of Black Hills' auriferous-deposits, 574, 575, 577 *et seq.*; of Black Hills tin, 596.
York, Walker County, Ala., coal-mines at, 210.

Zalinski dynamite gun, Fort Hamilton, New York harbor, visited [xliv].
Zinc in Ontario, Canada [294].
Zinc-dust, reducing action of on ferric solution, 411 *et seq.*
Zinc-oxide: Used in determination of arsenic, 77; In flue-dust at Low Moor furnace, 130.
Zircon in the Black Hills, Dakota [593].
Zwitter of Altenberg compared with the tin-bearing rock of the Black Hills, 594.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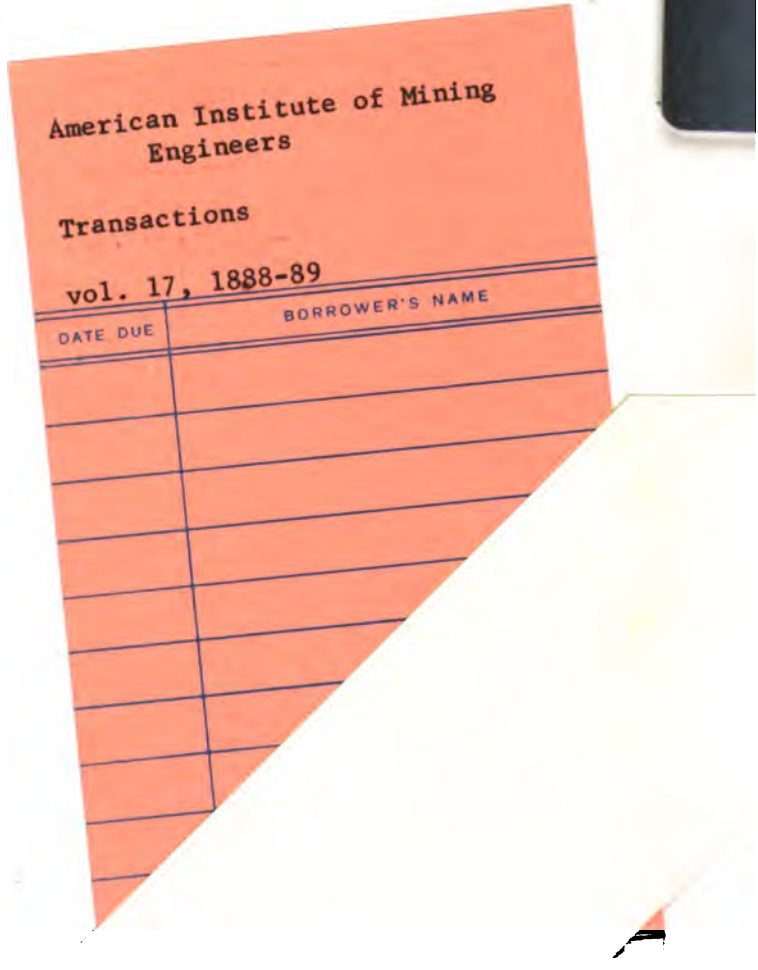
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Transactions
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